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METHODOLOGY FOR CHARACTERISING THE EFFICACY OF BLASTING IN OPEN-PIT MINES;

VIDEO AND IMAGE ANALYSIS

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God is great.

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“*Without leaps of imagination we lose the excitement of possibilities*.”

Gloria Steinem

“*To know that we know what we know, and to know that we do not know what we do not know, that is true knowledge*.”

Nicolaus Copernicus

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ABSTRACT

Rock Blasting is a very fundamental and vital operation in mining including open-pit mining as it serves to fragment rock and expose the desired mineral from the waste rock.

There are different problems which engineers may encounter during rock blasting both in underground and surface mining, the research project focuses on specific problems that engineers may face in open-pit mining and which could be viewed from a recorded video of the blasting operation, for example; excessive air-blasts, fly-rocks, excessive dust, bad fragmentation and escape of explosive energy.

The research project firstly looks at the engineering theory of explosives, rock blasting, blast design and rock fragmentation mechanics in open-pit mining and then focuses on blasting problems which could be seen in video recordings of rock blasting operations with the intention of outlining in the end engineering solutions to the problems identified. Videos of recorded blasting operations have been edited by Photoshop CC Software to produce still images which highlight blasting problems. Videos were collected from YouTube and are from the United States, Canada and Australia. Video data has also been noted and is included in the Appendix of the dissertation, video data includes; video URL, date of upload on YouTube, duration of video, location of the blasting operation.

Secondly, the project examines a model for analysing the risk associated with misfire detonations and provides information on the current occupational injuries statistics together with the technical aspects of occupational safety and health for handling explosives used for rock blasting in open-pit mines.

Mine accident statistics are from the United States, Canada, the United Kingdom, Portugal, Australia and the European Union (EU).

KEYWORDS:

ROCK BLASTING, OPEN-PIT MINING, SAFETY, EXPLOSIVES, ROCK FRAGMENTATION MECHANICS, BLAST DESIGN

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ABBREVIATIONS

HSE – United Kingdom Health and Safety Executive

MSHA – United States Department of Labor; Mine Safety and Health Administration

EU – European Union

OSHA – United States Department of Labor; Occupational Safety and Health Administration

ESSEEM – European Shot-firer Standard Education for Enhanced Mobility

DGEG – Direcção-Geral de Energia e Geologia (Portuguese Geology and Energy Directorate)

PORDATA – Base de Dados Portugal Contemporâneo (Portuguese Government Database)

US – United States

UK – United Kingdom

ONS – United Kingdom Office for National Statistics

CCOHS – The Canadian Centre for Occupational Health and Safety

MAC – The Mining Association of Canada

EU-OHSA – European Agency for Safety and Health at Work

ESAW – European Statistics on Accidents at Work

ABS – Australian Bureau of Statistics

CME – The Chamber of Minerals and Energy of Western Australia

GDP – Gross Domestic Product

WA – Western Australia

EU-27 – 27 Member States of the EU before Croatia joined on 1st July 2013

ENGINEERING SYMBOLS

λ – 1st Lamé Parameter (value depends on type of rock) (MPa)

$\boldsymbol{\beta}$ – S-Wave Velocity (m/s)

$\boldsymbol{v_p}$ – Velocity of P-wave (m/s)

$\boldsymbol{E_s}$ – Energy of Blasting Charge (kg m^2/s^2)

$\boldsymbol{V_s}$ – Volume of explosives for the whole blasting operation (m^3)

$\boldsymbol{P_{Z0}}$ – Effective Detonation Pressure of Explosive per unit Volume (Pa = kg/m s^2)

$\boldsymbol{\lambda_s}$ – Spacing (m)

$\boldsymbol{\xi}$ – Fill Factor (the volume of explosive vs. volume of the blast-hole)

$\boldsymbol{V_{s0}}$ – Volume of explosive per blasted unit volume (m^3)

$\boldsymbol{w'}$ – Average Blasted Burden (m)

$\boldsymbol{a'_B}$ – Average Blasted Blast-hole Distance (m)

$\boldsymbol{l_{B0}}$ – Unit Length of Blast-hole Constant = 1 m

$\boldsymbol{P_h(t)}$ – Hydraulic Shock Wave Pressure (kPa)

$\boldsymbol{PAP_x}$ – Explosive absolute weight strength (cal/g)

$\boldsymbol{\rho_e}$ – Density of Explosive (g/cm^3)

$\boldsymbol{U_p}$ – Particle Velocity (m/s)

$\boldsymbol{w_e}$ – Work of Expansion (kgm)

RQD – Rock Quality Designation (%)

PPV – Peak Particle Velocity (ips – Inches per second)

$\boldsymbol{\rho_r}$ – Rock Density (g/cm^3)

$\boldsymbol{f}$ – Frequency (Hz)

$\boldsymbol{P_{ZM}}$ – Effective Detonation Pressure for the whole blasting system (kPa)

μ – Shear Modulus – 2nd Lamé Parameter sometimes given as G (Rigidity Modulus) (MPa)

$\boldsymbol{E_{Kin_M}}$ – Kinetic Energy of the Muck Pile (J = kg m^2/s^2)

$\boldsymbol{m_M}$ – Muck Pile Mass (kg)

$\boldsymbol{c_M}$ – Blow-out Velocity of Muck Pile (m/s)

$\boldsymbol{g}$ – Acceleration due to Gravity (9.81 m/s^2)

$\Delta\boldsymbol{h_w}$ – Bench or Throw Height (m)

$\boldsymbol{E_{Pot_M}}$ – Potential Energy of the Muck Pile (J = kg m^2/s^2)

$\boldsymbol{E_{dyn}}$ – Dynamic Energy (J = kg m^2/s^2)

$\boldsymbol{E_{qstat}}$ – Quasi Static Energy (J = kg m^2/s^2)

$\boldsymbol{ppv_{max}}$ – Maximum Peak Particle Velocity (m/s)

VD – Detonation Velocity (m/s)

PD – Detonation Pressure (kPa)

PE – Thermochemical Pressure (kPa)

ET – Strain Energy of Explosive (kJ)

EB – Bubble Energy of Explosive also known as Gas Energy (kJ)

VC – Wave Propagation Velocity through the Rock (m/s)

$\boldsymbol{P_{Z0}}$ – Effective Detonation Pressure of Explosive per unit Volume (Pa = kg/m s^2)

1

INTRODUCTION

“*All truths are easy to understand once they are discovered; the point is to discover them.*”

Galileo Galilei

1.1 Objectives and Background

Open-pit mining as opposed to underground mining involves excavating rock and/or mineral from an open borrow on the surface, and because of the need for excavation, rock blasting plays a very vital and crucial role. Materials mined by open-pit mining include; coal, uranium, limestone, gypsum, granite, metal ores (like copper, iron, gold, tungsten, silver) and many more.

There is need to identify blasting problems which may occur during open-pit mining operations because they may affect the rock fragmentation efficiency, may cause accidents, may cause damage to the environment and one way of identifying blasting problems is by looking into and analysing video recordings of the blasting operation.

Today, mining engineers are well equipped with high tech computation methods which aid in finding solutions to different blasting problems in surface mining but also in underground mining.

The research project looks at problems, good practices and engineering solutions with the aid of still images produced from video recordings of blasting operations in the open-pit mining industry.

1.2 Methodology

At first there was an intensive research done by looking at recent papers and publications on the engineering principles and theory of rock blasting, rock fragmentation mechanics, explosives properties, blast design in open-pit mines and good blasting practices in open-pit mining.

Videos of rock blasting operations in open-pit mines in the United States, Canada and Australia were then downloaded from YouTube, the three countries have been chosen because on YouTube, a lot of videos on rock blasting in open-pit mines are from these three countries and they do provide a wide variety.

Each video was recorded in a table that is put in the Appendix of this dissertation, information on duration of video, URL address of video, date of upload on YouTube, filming location on each videos was collected.

Photoshop CC software was then used to edit the videos and produce still images which highlight blasting problems as well as good blasting practices.

MATLAB has also been used to calculate the effect of increasing the Explosive Charge Mass per Delay on Blasted Ore Throw using a Throw Trajectory Model developed by Workman et al in 1994.

2

HISTORY OF OPEN-PIT MINING AND ROCK BLASTING

"*A man may imagine things are false, but he can only understand things that are true, for if things be false, the apprehension of them is not understanding*."

Sir Isaac Newton

2.1 The History

What would be considered as modern open-pit mining dates back to the 16th century (*Montrie, 2003*) where the mined material was used for making tools and weapons and as building material but there have been open-pit mines since the beginning of human civilization.

Ancient Egyptians used to mine Malachite ($Cu_2CO_3(OH)_2$) as early as before 2494 BC, during that time there were also quarries for Turquoise ($CuAl_6(PO_4)_4(OH)_8.4H_2O$) and Copper (Cu), the ancient Greek Author, *Diodorus Siculus* describes these mines and he mentions how ancient Egyptians used fire-setting to break down hard rock. *Fire-setting* was also used by the Romans to fragment rock, basically, fire was set on the rock face in order to heat up the rock, the rock was then doused with a stream of water this created thermal shock which then broke down the rock. Some ancient mines include; *Maadi* in the Egyptian Empire, *Laurium* in Ancient Greece and there were also many ancient surface mines in ancient Spain during the Roman Empire.

Explosives were first used in mining for rock blasting in 1627 (*Buffington Gary L., 2000*), where gunpowder was used to fragment rock in what used to be a Hungarian town of *Banská Štiavnica*, now a Slovakian town.

The first recorded use of commercial explosives was the use black powder by the Chinese before the 13th Century (*Du Pont*). **Figure 1** shows the history of using Black Powder;

The Chronology of Black Powder

13th Century	First mention of saltpeter in writings of Abd Allah (Arabian), who called it "Chinese Snow."
13th Century	Description of "Roman Candles" in Chinese annals of the Sung dynasty.
1242	Friar Roger Bacon wrote the formula for black powder.
About 1300	Berthold Schwarz reputed first to use black powder as a firearm propellant.
1627	First documentary proof of black powder's use in mines by Kasper Weindl at Royal Mines of Schemnitz at Ober-Biberstollen, Hungary.
1675	Powder mill erected at Milton, Mass.
1689	Black powder used in the tin mines of Cornwall, England.
1696	Black powder used on Albula road construction in Switzerland.
1705	Probable black powder use at copper mines in Simsbury, Connecticut. Converted by blasting to the Newgate Prison in 1773.
1804	Eleuthère Irénée du Pont began commercial production of black powder in Wilmington, Delaware.
1857	Lammot du Pont substituted Chilean saltpeter (sodium nitrate) for more expensive potassium nitrate in black powder ("B" Blasting Powder).
1917	World War I — Black powder consumption in the United States reached a high of 277,118,525 pounds.
1930-40	Numerous black powder mills ceased production due to lack of market.
1973	Du Pont withdrew from the manufacture of black powder, selling their last plant to Gearhart-Owen Industries.

Figure 1 History of Black Powder

Source: Blaster's Handbook, Du Pont

Figure 2 shows the history of using Dynamite;

The Chronology of Dynamites

1846	Ascanio Sobrero discovered nitroglycerin.
1861	Alfred Nobel built small nitroglycerin plant near Stockholm at Heleneborg, Sweden.
1863	Alfred Nobel patented black powder, nitroglycerin mixture.
1866	Alfred Nobel mixed nitroglycerin with kieselguhr to make dynamite. George Mowbray built nitroglycerin plant at Titus, Pennsylvania.
1875	Alfred Nobel discovered "blasting gelatin" by dissolving nitrocellulose in nitroglycerin and introduced "gelatin dynamite."
1880	Du Pont formed the Repauno Chemical Company in New Jersey to manufacture dynamite.
1880's	Permissible dynamites investigated for coal mines in Europe.
1908	U. S. Geological Survey required dynamites be tested for permissibility prior to use in coal mines.
1925	Ethylene glycol dinitrate with nitroglycerin solved problem of dynamite freezing.
1936	Biazzi Continuous Process for nitroglycerin production demonstrated in Europe.
1950's	Ammonium nitrate, combined with various combustibles, began to replace large quantities of nitroglycerin dynamite. Water gels commercialized.
1974	Du Pont announced plans to phase out of dynamite business in favor of a new water gel explosive, "Tovex".

Figure 2 History of Dynamite

Source: Blaster's Handbook, Du Pont

Below is a timeline of the history of explosive use and development until 1995;

1200 A.D.

Arabian author Abd Allah records use of saltpeter as main ingredient of black powder.

1242

English Friar Roger Bacon publishes gunpowder formula.

1380

German Franciscan Monk, Berthold Schwarts developed gunpowder and its use in guns.

1745

Doctor Watson of British Royal Society explodes black powder with an electric spark.

1750

American inventor Benjamin Franklin encases and compresses powder in cartridges.

1831

William Bickford of Cornwall, England invents Safety Fuse.

1846

Italian chemist Ascanio Sobrero discovers nitroglycerine.

1863

Wilbrand invents Trinitrotoluene (TNT).

1864

Swedish inventor Alfred Nobel develops first detonating blasting cap.

1866

Swedish chemist Alfred Nobel invents dynamite by mixing kieselguhr with nitroglycerine.

1875

Nobel patents blasting gelatine.

1884

Ammonium Nitrate (AN) becomes widely used in dynamite formulations.

1885

Two-component explosives used in New York Harbor.

1888

Nobel invents Ballistite, a dense smokeless powder.

1885

Permitted explosives officially recognized in Europe.

1902

Detonating cord introduced in Europe.

1907

Consumption of black powder in U.S. more than 287 million pounds.

1917

U.S. Explosives Act sets regulations for purchases.

1921

U.S. National Academy of Sciences studies Ammonium Nitrate (AN) after explosion in Oppau, Germany.

1924

Largest industrial blast to date in U.S. fired at California Blue Diamond quarry using 328,000 lbs. of dynamite 1924.

Mid 1920’s

Liquid Oxygen based explosives commercialized in U.S.

1931

Fiberboard cases approved for dynamite shipping.

1938

Modern PETN-filled fabric-covered detonating cord introduced in U.S.

1939

U.S. Bureau of Mines begins work on vibration standards.
Modern plastic explosives invented during WWII.

1946

Short interval millisecond delay electric blasting caps introduced.

1950's.

High-speed photography for blast analysis introduced.

Late 1950's

Prilled AN fuel mixture begins to replace dynamite.
Bulk trucks and loaders developed.

1969

Emulsion explosives introduced.

Late 1980's

Electronic delay detonators (EDD's) introduced.

1992

Explosives used to extinguish most of 700 Kuwaiti oil well fires after Gulf War.

1995

One dynamite plant still operating in the United States.
Australia's largest shot ever, 1.25 million pounds of explosives at Ord River project.

(Source of History: Du Pont Blasters' Handbook)

3

INTRODUCTION TO EXPLOSIVES

3.1 Type of Explosives

An *explosive* can be defined as any material or device which can produce a sudden outburst of gas, applying a high impulsive loading to the surrounding medium (*Cook, 1958*).

Chemical explosives are normally used in mining. There are two categories of Industrial Chemical Explosives which include;

a) ***Deflagrating Explosives***: these type of explosives also called *Low Explosives*, burn relatively slow and produce a relatively low blast-hole pressure. E.g. Gunpowder.

b) ***Detonating Explosives***: these are also called *High Explosives* and are characterized by a super-acoustic reaction rate and produce high blast-hole pressure. E.g. Ammonium, Nitrate Fuel-Oil (ANFO).

Detonating Explosives (High Explosives) are further classified into three categories based on their sensitivity to the ease of initiation and detonation, and this class consists of;

1. Primary Detonating Explosives

These explosives are initiated by a spark or impact, they are very unstable compounds and therefore are used industrially only in initiating devices such as blasting caps as the top charge. Examples in this class include; Lead Azide ($Pb(N_3)_2$), Lead Styphnate ($C_6HN_3O_8Pb$) and Mercury Fulminate ($Hg(CNO)_2$.

2. Secondary Detonating Explosives

Secondary detonating explosives require the use of a blasting cap for practical initiation and in some cases may require an ancillary booster charge. Explosives in this class are formulated from chemicals like Nitro-glycerine (NG), Penta-erythro-tetranitrate (PETN) or Ethylene-glycol-dinitrate (EGDN) mixed with other explosive materials and stabilizing agents.

3. Tertiary Detonating Explosives

Tertiary detonating explosives are not sensitive to a standard No. 6 strength blasting cap and most of the detonating explosives in this category are dry blasting agents (DBAs) or slurry explosives and blasting agents. Examples are ANFO, Explogel, Seismogel and ALANFO.

In the past most high explosives were manufactured from organic nitrates mixed with both organic and inorganic chemicals in order to produce chemically and mechanically stable materials known as *Gelignites* and *Dynamites*. To date, the proportion of organic nitrates in high explosives has been reduced, for example, by their partial replacement with chemicals like ammonium nitrate (AN). Ammonium Nitrate Fuel-Oil (ANFO) is a good example of a blasting agent which usually comprise of 94% AN - 6% fuel-oil. ANFO is hence an oxygen-balanced mixture of an oxidizer and a fuel and it means by achieving a close admixture of these two compounds, the material can be detonated and would yield CO_2, NO_2 and H_2O (*Brady B. H. G., 2005*).

During detonation in a blast-hole which is an example of an explosive column, a chemical reaction front passes through the column being driven by the products of the reaction itself at a velocity above the speed of sound (super-sonic), this velocity is called *detonation velocity*, passage of this chemical reaction front causes a rise in pressure in the explosive which results in *detonation pressure* with a magnitude of 10^9 Pa and above.

3.1.1 Examples of Industrial Explosives

a) **DRY BLASTING AGENTS**

Dry blasting agents are explosives which are not cap sensitive and do not include water in their composition. Ammonium Nitrate (NH_4NO_3) is present in all dry blasting agents. Examples of dry blasting agents include; ANFO and Aluminised Ammonium Nitrate Fuel-Oil (ALANFO).

The density of ANFO is low, hence, Aluminium powder is added to ANFO to produce ALANFO in order to increase the density of ANFO thereby increasing its energy per metre of column length, this is beneficial when blasting strong and massive rocks and also in high cost drilling operations.

There are standards of adding Aluminium to ANFO;

The Aluminium powder should be 100% in size between 20 and 150 mesh and should be more than 94% pure.

b) SLURRIES

Slurries are explosives which are based on saturated aqueous solutions of Ammonium Nitrate and oxidisers like Sodium Nitrate ($NaNO_3$), Calcium Nitrate ($Ca\ (NO_3)_2$).

Examples include; Explogel V4, Explogel Pillowpak, Tovex also known as Seismogel (it contains a mixture of Ammonium Nitrate and Methyl-Ammonium Nitrate (MAN)).

c) **EMULSIONS**

Emulsions are relatively new in the explosives industry compared to traditional explosives, the development of emulsion explosives started in the sixties (*Jimeno Carlos L., 1995*).

Emulsion explosives have the same properties as Slurry explosives but with high strength and water resistance. Examples include; RIOMEX, RIONEX.

d) **HEAVY ANFO**

Heavy ANFO is produced by combining ANFO with a base emulsion in order to increase the explosive energy.

Engineers are always investigating different methods of obtaining the maximum energy from ANFO like using high energy liquid fuels; methanol, nitro-paraffins and nitro-propane in combination with Ammonium Nitrate.

Heavy ANFO produces more energy, has better sensitivity, has high water resistance and provides the possibility of charging with variation in energy along the blast-hole length when compared with conventional ANFO.

e) **GELATINE DYNAMITES**

Gelatine dynamites are explosives which have Nitro-glycerine (NG) and Nitro-cellulose (NC) percentage combination (NG-NC) between 30 to 35% and 65 to 70% oxidisers like NH_4NO_3.

The risk involved when working with gelatine dynamites is very high because of its high sensitivity and sensitivity to subsonic stimuli, hence, maximum care should be taken when manufacturing and transporting gelatine dynamite explosives as the probability of accidental explosions is high.

f) **GRANULAR DYNAMITES**

Granular dynamites have a mixture of NG below 15% and have a granular powdery consistency.

g) **BLACK POWDER EXPLOSIVES**

At present black powders used in mining have a composition of 75% Potassium Nitrate (KNO_3), 10% Sulphur (S) and 15% Carbon (C). The strength of black powders is roughly 28% that of gelatine dynamites.

Tables 1 and **2** summarise examples of industrial explosives commonly used in mining operations;

Table 1 Summary of Common Industrial Explosives

	Commercial denomination	Relative strength (%)	Density (g/cm^3)	Velocity of detonation (m/s)	Specific energy (kgm/kg)	Water resistance	Applications
ANFOS	Nagolita	65	0.80	2.000	94.400	Bad	Opencast operations, in the absence of water. Column charging.
	Alnafo	75	0.80	3.000	96.100	Bad	Blasting of soft and medium rocks.
	Naurita	65	0.80	2.000	94.320	Bad	Designed for blastholes with high temperatures.
Water gels	Riogel 2	85	1.17	4.500	94.801	Excellent	Blasting in presence of water. Base charge in open cast and quarrying. Tunelling and shaft sinking.
	Riogur	73	1.1	3.300	88.813	Excellent	Profilling and presplitting blasts.
Emulsions	RIOMEX E20-24 (Cartridged) Series E20-E24	103-145*	1.15	4.500-5.200	–	Excellent	Underground and surface blasting. Excavation of ditches, trenches and general civil works. Wet blastholes.
	RIONEX V2 (Bulk) Series V20-V24	109-154*	1.23	5.000-5.400	–	Excellent	Open pit mining. Large quarries. Public works. Able to use with automatic loading sistems.

*RBS = Relative Bulk strength % (RBS ANFO = 100).

Table 2 continuation of Table 1

	Commercial denomination	Relative strength (%)	Density (g/cm^3)	Velocity of detonation (m/s)	Specific energy (kgm/kg)	Water resistance	Aplications
Gelatinous	GOMA 1ED	90	1.49	6.000	104.158	Very good	Blasting of very hard rocks.
	GOMA 1-AGV	80	1.55	7.000	96.800	Very good	Blasting under high pressures. Seismic prospections.
	GOMA 2EC	85	1.40	5.250	102.400	Good	Blasting unnder presence of water. Base charge: combined with blasting agents. Hard rocks. Underground and surface blasts.
Powder explosives	AMONITA 2I	77	0.95	3.000	87.500	Poor	Blasting of soft to medium hard rocks.
	LIGAMITA 1	77	1.10	3.300	86.400	Poor	Blasting of soft to medium hard rocks. Used in underground mining and quarrying as well as in underground operations.
Permitted explosives	EXPLOSIVO DE SEGURIDAD 9	45	1.50	4.500	49.150	Good	Used in hard rocks and coal blasting operations.
	EXPLOSIVO DE SEGURIDAD 20 SR	40	1.15	2.500	36.650	Poor	Used in underground coal blasting operations.
	EXPLOSIVO DE SEGURIDAD 30 SR	35	1.10	2.000	30.190	Poor	Used in underground coal blasting operations and soft rocks.

Source: Jimeno Carlos L., 1995

3.2 The Strength and Energy of Explosives

One way to estimate the strength of an explosive is by estimating its free-energy output from the thermodynamics of its detonation reaction, this is termed as the *Absolute Strength Value* (ASV), and the ASV of an explosive is the free energy output (Joules) per 0.1 kg of explosive.

$$ASV = \frac{Free\ Energy\ Output\ (J)}{0.1\ (kg) of\ Explosive} \tag{3.10}$$

However, the ASV is not a complete estimate of the potential performance of the explosive on a given rock because there is partitioning of explosive energy produced by the explosive between *shock energy* and *bubble energy*. A closer estimate is called the *Brisance* which is the potential shattering action of an explosive (*Brown E. T., 2005*).

Brisance is directly related to the detonation pressure and the detonation pressure is itself related to the detonation velocity. Brisance of an explosive is an approximation and can be estimated as follows;

$$Brisance \approx \rho D^2 \tag{3.11}$$

ρ – Density of explosive

D – Detonation Velocity of a given explosive

High Brisance Explosives have detonation velocities higher than 5000 m/s and Low Brisance Explosives have detonation velocities lower than 2500 m/s. For an explosive to be effective and perform successfully in a given rock mass, its properties should be compatible with the properties of the rock mass. **Figure 3** shows an empirical correlation of preferred type of explosive for a range of rock materials and rock mechanical properties.

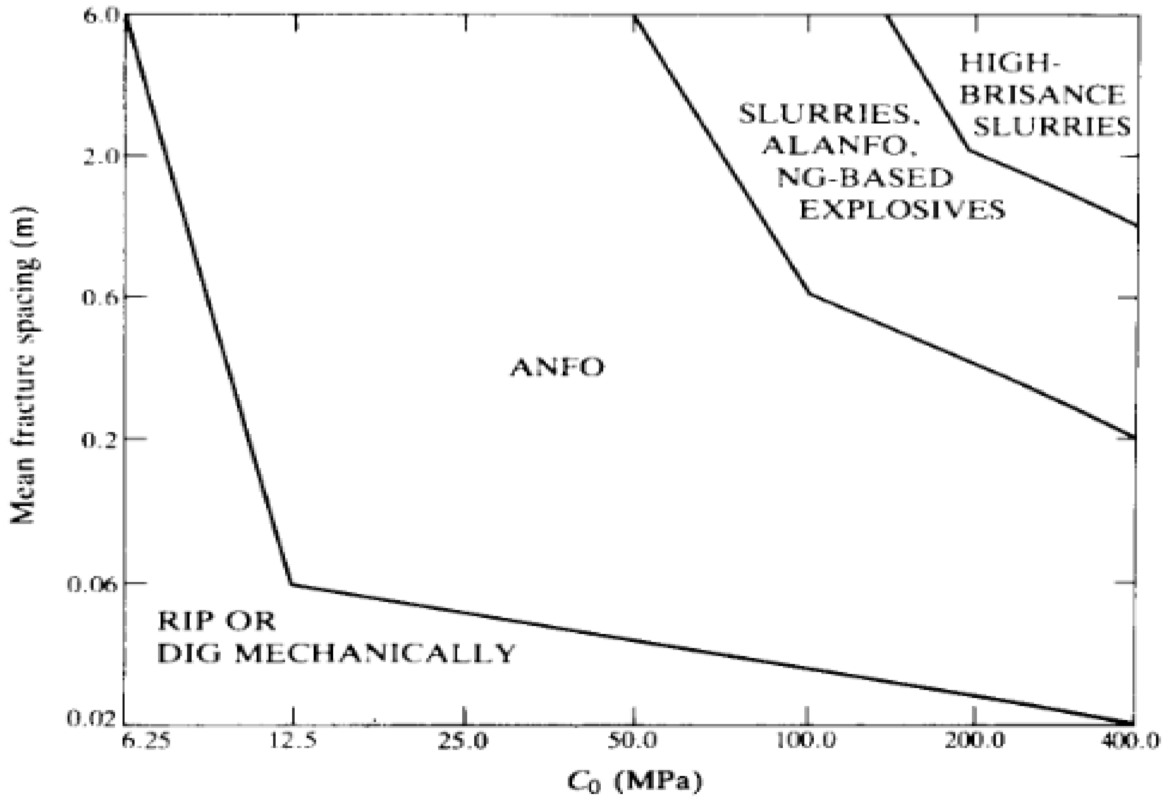

Figure 3 Empirical Correlation Rock Properties and Type of Explosive

Source: Brady B. H. G., Brown E. T. 2005

The strength and energy aspect of an explosive is looked upon from an industrial application point of view because the strength of an explosive directly defines how much energy that explosive can give in order produce the desired mechanical effects on a rock.

There are many equations and empirical formulas available to engineers to quantify the strength of explosives used for rock blasting, two such equations have already been highlighted in equations **3.10** and **3.11** by using the ASV and the Brisance respectively, another important test that yields other vital equations and empirical formulas to quantify the energy that an explosive can provide for mechanical use is the *Underwater Test;* a test that was carried by Cole in the 1960s, and is considered in the mining and explosives industry as the most complete test (*Jimeno Carlos L., 1995*) because it accurately depicts what happens when an explosive is detonated in the blast-hole having the same charge geometries exactly as in the blast-hole.

The underwater test provides two important parameters; the *Strain Energy of Explosive* (ET) and the *Bubble Energy of Explosive or Gas Energy* (EB) and allows each of these two parameters to be calculated separately. The ET is the explosive energy linked to the strain wave while the EB is the energy of the detonating gases.

Hence ET is calculated from an Underwater Test as follows using Blanc's equation (1984);

$$ET = \frac{4\pi DS^2}{\rho_0 VH} . \int_{t_1}^{t_2} p_h(t)dt \qquad (3.12)$$

$P_h(t)$ – Hydraulic Shock Wave Pressure

DS – Distance of the explosive charge from capacitor

ρ_0 – Volumetric mass of water

The Gas Energy (EB) is calculated using the Willis formula as follows;

$$EB = K\left(\frac{P_h^5}{\rho_0^3}\right)^{0.5} . t_B^3 \quad (3.13)$$

K – Explosive constant

$P_h(t)$ – Total Pressure at the location of submerged charge (atmospheric pressure + hydrostatic pressure)

ρ_0 – Volumetric mass of water

t_B – The 1st oscillation pseudo-period of explosive gas bubble after detonation

Figure 4 shows the experimental setup for the Underwater Test; (**Jimeno Carlos L., 1995**)

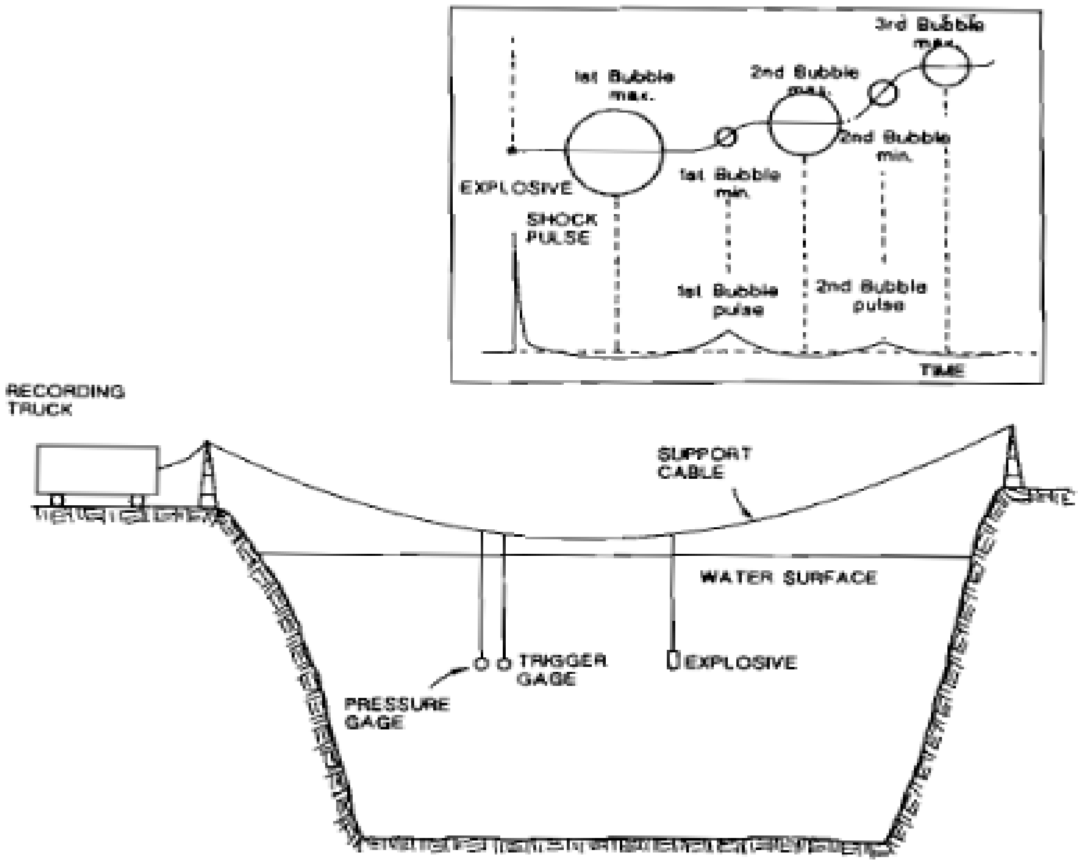

Figure 4 Underwater Blasting Test

However, practically there are also very vital empirical equations developed through the use of different experimental and field data which engineers use to quantify the explosive energy and strength of explosives.

The Swedish Empirical equation is used to calculate the *Relative Weight Strength* (PRP) of an explosive;

$$PRP = \frac{5}{6}.\frac{Q_e}{Q_0} + \frac{1}{6}.\frac{VG}{VG_0} \tag{3.14}$$

PRP – Relative Weight Strength of Explosive

Q_0 – Heat of explosion of 1 kg of explosive LFB (5 MJ/kg) under normal pressure and temperature conditions

Q_e – Heat of the explosion of 1 kg of explosive used

VG_0 – Volume of gases released by 1 kg of explosive LFB (0.85 m^3/kg)

VG – Volume of gases liberated by the used explosive

ANFO has Q_e of 3.92 MJ/kg and a VG of 0.973 m^3/kg, ANFO is one of the explosives commonly used in the mining industry for rock blasting.

PRP can also be calculated as follows;

$$PRP = \left(\frac{\rho_e.VD^2}{\rho_0.VD_0^2}\right)^{\frac{1}{3}} \tag{3.15}$$

ρ_e – Density of explosive (g/cm^3), VD – detonation velocity (m/s)

ρ_0 , VD_0 are for the standard explosive used in the calculation.

Paddock's empirical formula (1987) is used to quantify the explosive *Strength Factor* (FP) as follows;

$$FP = PAP_x . VD . \rho_e \tag{3.16}$$

FP – Strength Factor

PAP_x – Explosive absolute weight strength (cal/g)

VD – Detonation Velocity (m/s)

ρ_e – Density of Explosive (g/cm^3)

In most cases ANFO is taken as the standard explosive and when this is the case, the value of PAP_{ANFO} = 890 cal/g.

This leads to a relationship between the Relative Weight Strength (PRP) and the Explosive Absolute Weight Strength (PAP_x) as shown in **equation 3.17**;

$$PRP = \frac{PAP_x}{PAP_{ANFO}} \tag{3.17}$$

Hence, when ANFO is used as the standard explosive, using equation 3.8, the engineer can calculate the PRP of the explosive if the PAP_x value known, and vice versa, he can also calculate the $PAP_x$of a given explosive if the PRP value is known.

3.3 The Physics of Explosive Detonation

The most important objective when using explosives for rock fragmentation in mining is to ensure that the chemically concentrated energy from the explosive is placed properly and is of sufficient magnitude so that when it is liberated it should be controlled in space and time in order to achieve the desired rock fragmentation level.

In mining operations, the explosion mechanism of interest is chemical explosion but there are also other explosion mechanisms which are not applied in mining for rock breakage these are Nuclear (fission and fusion) and Mechanical. An example of a mechanical explosion mechanism occurs in thermodynamic explosives (e.g. boiling water in a closed vessel until the vessel explodes, putting water in an enclosed metallic pipe and then cooling the pipe with liquid nitrogen until the enclosed pipe explodes) and structural explosives (e.g. the shattering and explosion of a faulty flywheel due to its own stored kinetic energy).

In this dissertation only chemical explosion mechanisms are discussed as they are the ones that are applied to fragment rock in the mining industry.

3.3.1 Detonation Phase

Explosive detonation is a physicochemical process that is characterised by the high reaction velocity and the formation of huge quantities of gaseous products at elevated temperatures resulting in expansive force of great magnitude (*Jimeno Emilio L., 1995*).

There is a *shock wave* generated during the detonation of an explosive as the first gasified molecules attain high speed and do not lose heat to the surrounding charge zone through conductivity but rather transmit the heat by shock, deforming it in the process and then using it to generate new gases, this process repeats and it is the repetition of this process that generates shock waves.

There are different ways in which the initiation energy can be supplied, however, this depends on the type of the explosive being used; In Low Explosives (e.g. Gunpowder) the energy of the flame is enough but in High Explosives (e.g. Nitro-glycerine) there is need for a shock wave type of energy for initiation. After initiation, a shock or pressure wave is generated and is propagated through the explosives' own mass. The shock or pressure wave carries the energy that is required to further activate molecules of the explosive mass and this initiates a chain reaction and as this happens the reactive explosive mass produces a huge quantity of very high temperature gases which generate a secondary pressure and if this pressure acts on the undetonated explosive mass, its effect is summed to the primary pressure/shock wave but if the secondary gas pressure does not act on the undetonated explosive mass then there is a slow deflagration which slows down the explosive reaction and causes an energy loss in the primary pressure wave which becomes incapable of providing further energy to the rest of the explosive mass and the detonation is stopped. **Figure 5** illustrates this aspect;

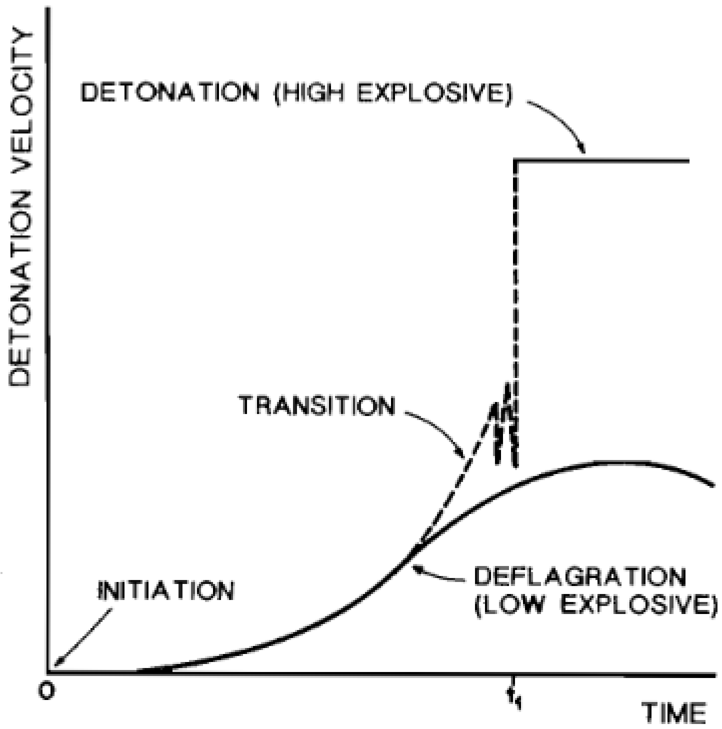

Figure 5 Detonation Process

Source: (Jimeno Carlos L., 1995)

The initial shock during detonation is responsible for activating the chemical transformation that follows and occurs in the reaction zone called the *Chapman-Jouget* (C-J) *plane*, **figure 6** shows the C-J plane of the reaction zone;

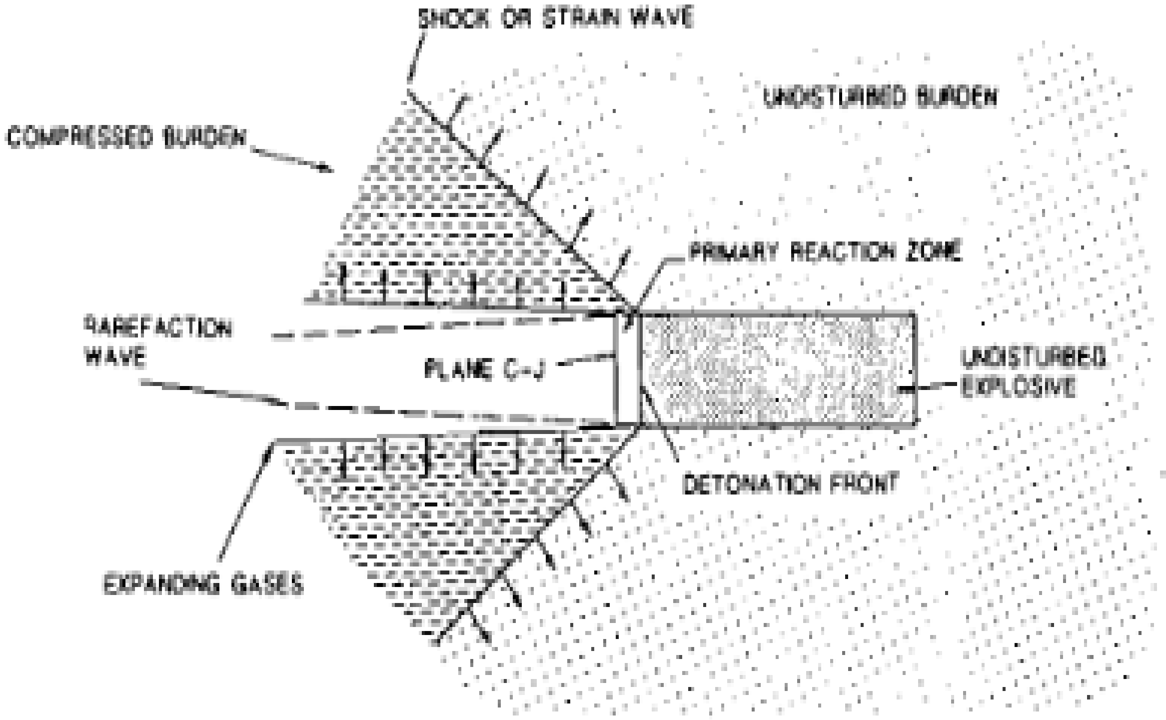

Figure 6 Explosive Charge Detonation Process

Source: Jimeno Carlos L., 1995

Deflagration is when the decomposition of the explosive material is propagated by the flame front by moving slowly through the explosive mass and is below supersonic, whereas in *Detonation*, the explosive material is propagated through shock and is supersonic.

Deflagration is common in low explosives e.g. gunpowder, fireworks

The C-J plane travels at a very high speed VD, and Melvin A. Cook using x-ray photographs was able to determine that the speed of explosion particles reaches a value of 0.25 times the detonation velocity (VD) and using this principle he was also able to relate the Detonation Pressure (PD) with detonation velocity (VD) as shown in equation 3.19;

$$PD = \rho_e . VD . U_p \tag{3.18}$$

PD – Detonation Pressure (kPa)

VD – Detonation Velocity (m/s)

ρ_e – Density of Explosive (g/cm^3)

U_p – Particle Velocity (m/s),

$U_p = 0.25 \times VD$, Hence;

$$PD = \frac{\rho_e . VD^2}{4} \tag{3.19}$$

The thermochemical pressure (PE) is about half the detonation pressure, hence;

$$PE = \frac{PD}{2} \tag{3.2}$$

The thermochemical pressure is responsible for carrying out the mechanical work on the given rock mass as it is the maximum pressure available (*Jimeno Emilio L., 1995*).

However when the explosive charge is very close to the wall of the rock of the blast-hole the thermochemical pressure equals the detonation pressure as in **equation 3.3**;

$$PE = PD \tag{3.3}$$

The detonation pressure (PD) can also be calculated using the detonation velocity and the density of the explosive because PD is basically the function of explosive density and the explosive detonation velocity (*Jimeno Emilio L., 1995*);

$$PD = 432 \times 10^{-6} \times \rho_e \left(\frac{VD^2}{1+0.8\rho_e}\right) \tag{3.4}$$

PD – Detonation Pressure (MPa)

ρ_e – Density of Explosive (g/cm^3)

VD – Detonation Velocity (m/s)

Commercial explosives have PD values ranging from 500 to 1500 MPa, PD values are significant because in hard and competent rocks the fragmentation process is easily achieved when high detonation explosives are used because they have high PD values.

Figure 7 illustrates the pressure profile during detonation in an explosive column;

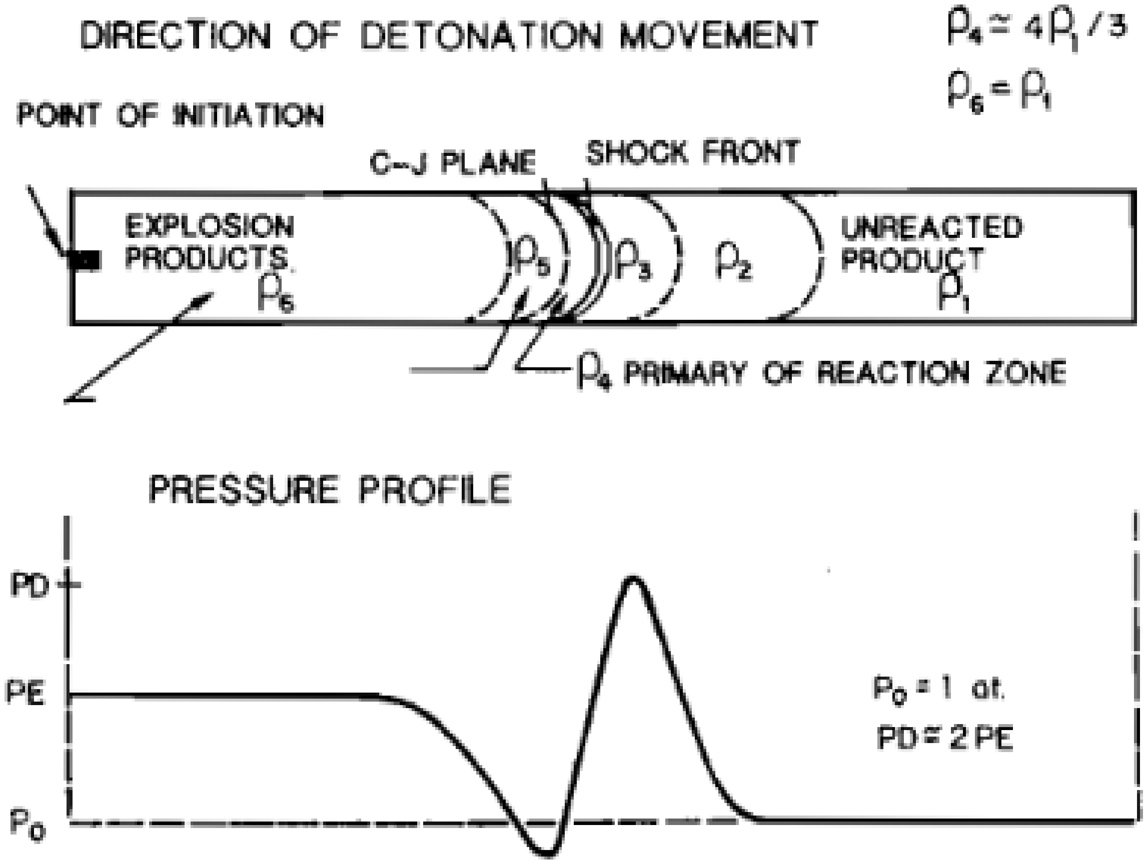

Figure 7 Explosive Detonation Pressure Profile

Source: Jimeno Carlos L., 1995

3.3.2 The Minimum Energy of an Explosive

The *Minimum Energy of an Explosive* is the quantity of work that the gaseous products of an explosion can do at a constant pressure of 1 atmosphere (atm).

When explosives are detonated the molecular volume occupied by gases of the explosion increases, sometimes increasing as high as 700% depending on the explosive used, while the resisting pressure stays constant (*Jimeno Emilio L., 1995*).

Therefore, the Work of Expansion can be calculated as;

$$w_e = P(V_2 - V_1) \tag{3.5}$$

w_e – Work of Expansion (kgm)

P = 1 atm (Resisting Pressure)

V_2 – Volume of gaseous products of explosion

V_1 – Volume of the explosive

And since the volume of the explosive (V_1) is negligible when compared with the volume of gases produced during the explosion (V_2), **equation 3.5** is simplified and becomes;

$$w_e = PV_2 \tag{3.6}$$

After w_e (quantity of work) is calculated for a given explosive it is taken as the ***minimum energy available*** for that explosive (*Jimeno Carlos L., 1995*).

3.3.3 Pressure of Explosion

The *Pressure of Explosion* during rock blasting is the force that acts on a given area during the explosion.

There is an empirical formula that engineers can use to calculate the Pressure of Explosion, the pressure of explosion is calculated in MPa when the density of the explosive is in kg/m^3 as follows;

$$P = f_s \left(\frac{\rho_e \times 10^{-1}}{1 - \alpha \rho_e} \right) \tag{3.7}$$

α – A constant which is calculated from the specific volume (V_s) (volume of the explosive or volume of the blast-hole between the explosives mass can be used);

$$\alpha = 0.92(1 - 1.07e^{-1.39V_s})$$

f_s – Specific Force

ρ_e – Density of Explosive (kg/m^3)

V_s – Specific volume (m^3)

4

SELECTION CRITERIA OF EXPLOSIVES IN MINING

4.1 The Criteria

Choosing the appropriate explosive to be used for rock blasting in mining operations is an important decision made by the engineer and becomes a significant part of blast design.

Below are elements engineers commonly use when selecting the type of explosive for rock blasting in mining operations;

a) **SAFETY CONSIDERATION**

Engineers are faced with the need to balance the sensitivity of an explosive and safety considerations, this need affects the selection process. As an example, Gelatine explosives are very sensitive and because of their high sensitivity if for some reason left-overs like breakages of the detonating cord are present in the working area where heavy machinery is used accidental detonation of the left-overs can occur and this presents a dangerous situation for mine operators working in the area. Safety should always come first when working with explosives.

b) **COST OF EXPLOSIVES**

The factor of cost is a very significant aspect in many engineering operations as a balance has to be struck between technical efficiency and the costs incurred, this also applies to the selection of explosives for use in rock blasting. The common practise is to choose the least costly explosives which can carry the amount of work required on a given rock mass.

c) **SUPPLY CHAIN**

Supply also affects the selection criteria, alternative explosives and accessories can be used when a particular explosive fails to be supplied due geographical restraints or other logistical problems.

d) CHARACTERISTICS OF ROCK

Geomechanical rock properties have to be considered when making a choice on which explosives to use for rock blasting because they affect the end results of the blasting process and are directly related to blast design parameters.

Figure 3 illustrates the relationship between rock properties and the type of explosives to be used.

Basically rocks can be classified into four types when assessing which explosives to use on a given rock mass (*Jimeno Carlos L., 1995*);

- Resistant Massive Rocks
- Highly Fissured Rocks
- Rocks which form Blocks
- Porous Rocks

Resistant Massive Rocks

In resistant massive rocks there are few fissures and weakness planes and this requires the creation of a large number of new surfaces by the explosive being used through its strain energy (ET) hence the recommended explosives to be used with these types of rocks are explosives with high density and high detonation velocity such as gelatine, slurries and emulsions.

Highly Fissured Rocks

Highly fissured rocks easily develop radial cracks when explosives with high ET are used but the cracks are interrupted due to the intersection by the pre-existing fissures in the rock mass.

Therefore, in these types of rocks it is recommended to use explosives with high gas energy (EB) like ANFO.

Rocks which form Blocks

Rocks that form blocks have a large spacing between discontinuities in the rock mass, this results into the formation of large-volume in-situ blocks and most often there are large boulders within the rock plastic matrix. Investigations have shown that in these types of rocks it is the geometry of the blast that mainly governs the fragmentation mechanism and only to a lesser amount does the property of explosives affect fragmentation in these rocks (*Jimeno Emilio L., 1995*). Explosives with balanced ET–EB relationship are recommended for blasting in rocks which form blocks. ALANFO and Heavy ANFO are the perfect examples.

Porous Rocks

In porous rocks it is the bubble energy (EB) that carries almost all of the work, this is so because of the great buffer-effect during the explosion where the strain energy (EB) is absorbed, hence, it is recommended to use explosives with low density and detonation velocity such as ANFO when blasting porous rocks.

It is also recommended to keep the blast gases within blast-holes for as long as possible, this can be done as follows;

- By controlling the stemming material and its height
- By using burdens with proper size
- By utilising bottom priming
- By decoupling charges and/or adding inert material e.g. ANFO mixed with expanded polystyrene beads (ANFOPS) in order to minimise the blast-hole pressure

e) **AMBIENT FACTORS**

The surrounding environment where rock blasting takes place also affects the selection of explosives to be used. The presence of water is such one ambient factor that is taken into consideration, this is because some explosives like ANFO do not detonate properly when the humidity level is above 10%. To try and resolve issues associated with water presence in blast-holes engineers can select another explosive that withstands high humid levels if applicable and this is done when dewatering is impossible, or if ANFO is still to be used, dewatering procedures like pumping using submersible pumps are applied and waterproof liners are also placed in the blast-hole. Explosives that can be used when the humidity and water content is high include; slurries, bulk water gels, gelatine and emulsions.

Atmospheric temperature is also another ambient factor that is considered because explosives which contain Nitro-glycerine freeze when the temperature goes below 8 ^{0}C, additives can be added to explosives which contain NG to lower their freezing point, this is particularly important when blasting in cold climates. Higher temperatures are also dangerous as most explosives become very sensitive and can detonate at high temperature, hence extra care should be taken when handling and using explosives in hot climates.

f) **QUANTITY OF ROCK TO BE BLASTED**

The volume of rock to be blasted is also taken into consideration when choosing the type of explosive for rock blasting. Calculations on the amount and type of explosive a company buys are based on the volume of excavation and blasting requirement together with work schedule.

As an example, in a large mining operation usually the amount of explosive used is taken in bulk form as this simplifies the use of mechanisation in charging from the transport units themselves this in turn reduces the cost of labour.

g) EXPLOSIVE AND ROCK IMPEDANCE MATCHING

Matching the explosive detonation impedance with the rock impedance is a great tool which helps in deciding which explosive to use on a given type of rock and more particularly so when there is a wider variation in explosive detonation velocities within a given zone of rock strengths. (**Figure 3**).

The theory of Impedance Matching recommends that the explosive impedance should as closer to the rock impedance as much as possible i.e. the impedance ratio (Z_r) must be very close to 1, (for example = 0.85) because theoretically the ideal scenario is for the explosive impedance to be equal to the rock impedance ($Z_r = 1$), this ensures maximum explosive energy distribution within the given rock during blasting (*Persson et al., 1994*).

Persson et al. came up with the Mathematical Impedance Matching relation as follows;

$$Z_r = \frac{\rho_e C_d}{\rho_r C_p}$$

Z_r – Impedance Ratio (Dimensionless)

ρ_e – Explosive Density (kg/m^3)

C_d – Explosive Detonation Velocity (m/s)

ρ_r – Rock Density (kg/m^3)

C_p – P–Wave Velocity (m/s)

4.2 Initiation Systems

Choosing the right initiation systems is as important as selecting the appropriate explosives for a particular rock blasting operation and the engineer should ensure that the chosen initiation systems are of high quality and are properly applied during the blasting operation, this in turn improves safety during blasting.

Modern technology has ensured that initiation devices are now much safer than they were in the 19th and 20th centuries.

Today, the use of non-electric initiation systems reduce the hazard of premature detonation that can be caused by stray current and radio energy. Modern initiation devices also lower the possibility of misfires by providing surplus firing energy (e.g. High Energy Blasting Devices).

The engineer should consider the following factors when selecting initiation devices;

a) Consult different manufacturers for information on their initiation devices
b) Geology of the mine site (Rock Mechanics and Geomechanical properties)
c) Blast design patterns to be used
d) Induced Vibrations (Peak Particle Velocity, Frequency, local and national legislation)
e) Desired degree of rock fragmentation
f) Explosives properties and characteristics
g) Blast-hole conditions (presence of water, voids, fractures, weathering degree)
h) The potential for an Air-blast (including local and national legislation)

According to the *US Department of the Interior; Office of Surface Mining Reclamation and Enforcement* an ***Initiation System*** is basically a system that provides the initial energy required to detonate an explosive during rock blasting.

In order to work, initiation systems require the following;

1) An initial energy source
2) Distribution network that delivers that energy to each blast-hole
3) An in-hole component responsible for initiating explosives that are detonator sensitive

There are two kinds of initiation systems; **electric** and **non-electric**.

Initiators can generally be classified as follows;

a) Blasting cap and Safety fuse systems
b) Electronic Systems
c) Non-Electric Systems
d) Electric Systems

High explosives require detonators to initiate their detonation. **Figure 8** shows a detonator;

Figure 8 A Detonator as a complete Initiation Device

Source: Best D., 2008

As can be seen in figure 12, a detonator contains the initiation signal transmitter (e.g. shock tubes, leg wires etc.) and an active part which is enclosed in a metal casing.

There are three kinds of detonators;

i) Instantaneous Detonators – Have no time delay
ii) Millisecond Delay Detonators – commonly used in surface mine blasting with delays up to 500 ms
iii) Long-Period Delay Detonators – delay of up to a couple of seconds

Figure 9 shows two types delay mechanisms; **Electronic** and **Pyrotechnic Delays**;

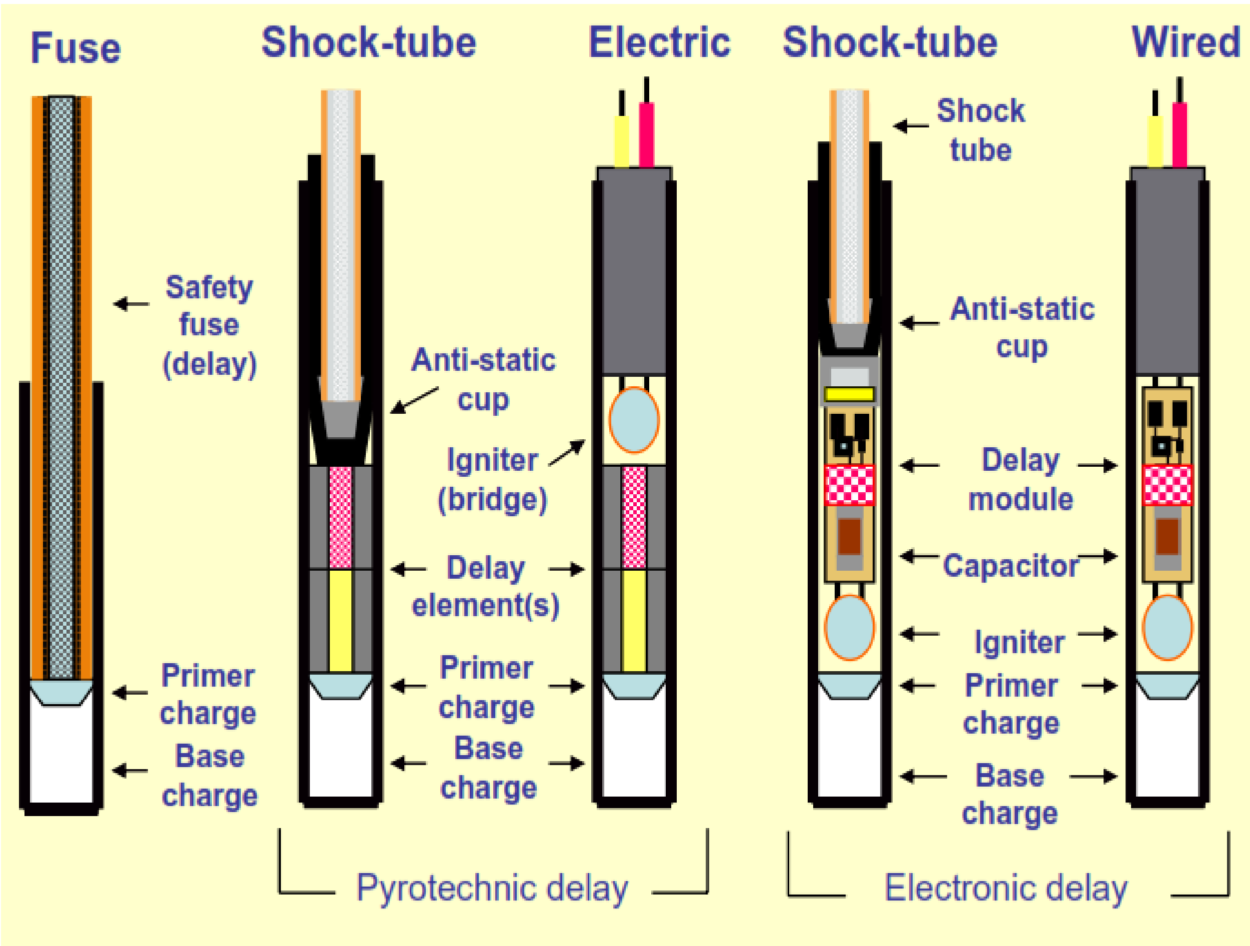

Figure 9 2 Types of Delay Mechanisms; Electronic and Pyrotechnic Delay

Source: Best D., 2008

It is highly recommended to use the detonating cord when the blasting operation will require extreme loading conditions and/or multiple priming. The detonating cord should not be used when the blast-holes have small diameters as this may damage the explosive during detonation.

Sensitive explosives should not be used together with a detonating cord due to the danger of premature initiation.

In places which are prone to stray currents, static electricity, radio frequencies, thunderstorms, electrical blasting systems should never be used due to the grave danger of accidental detonation, as an example, due to this reason electrical blasting systems are banned in the western United States. In such places, electronic systems, detonating cords or shock tube systems should be used.

Figure 11 Detonating Cord

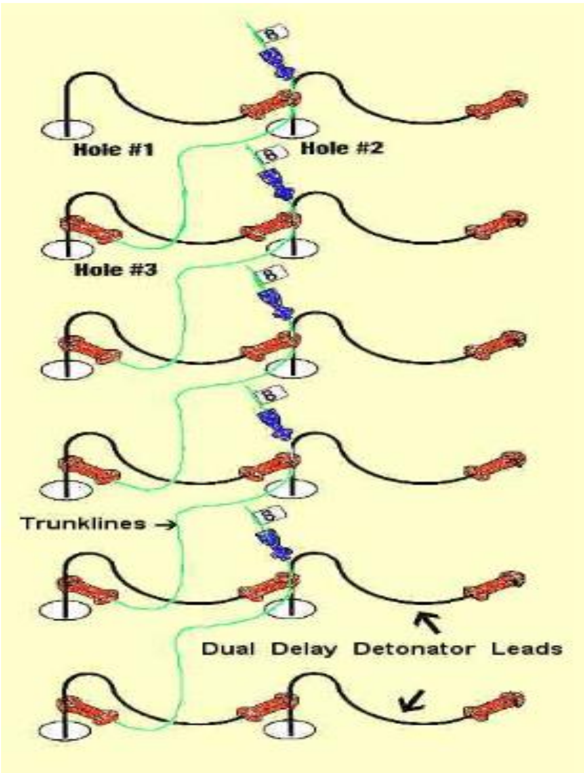

Figure 10 Shock Tube System

Source: Best D., 2008

Electronic Initiation Systems are now widely used in the mining industry and they have very good advantages over other systems which include; increased security and safety, high precision and accuracy, design flexibility, easy blast management.

Disadvantages of electronic initiation systems include; very expensive price compared to other initiators, electronic initiators are complex to operate and hence require further training.

Non-electric systems include shock tube detonators, detonating cords and the combination of the two and as discussed earlier there are very good alternatives to use in places prone to static currents, stray currents and radio frequencies but lightning can still cause an accidental detonation when non-electric initiators are used. The disadvantage of non-electric initiators is that they are very sensitive to mechanical impact and heat because they contain ignition and base charges.

Figure 12 shows a detonating cord set-up as can be applied at a surface mine site;

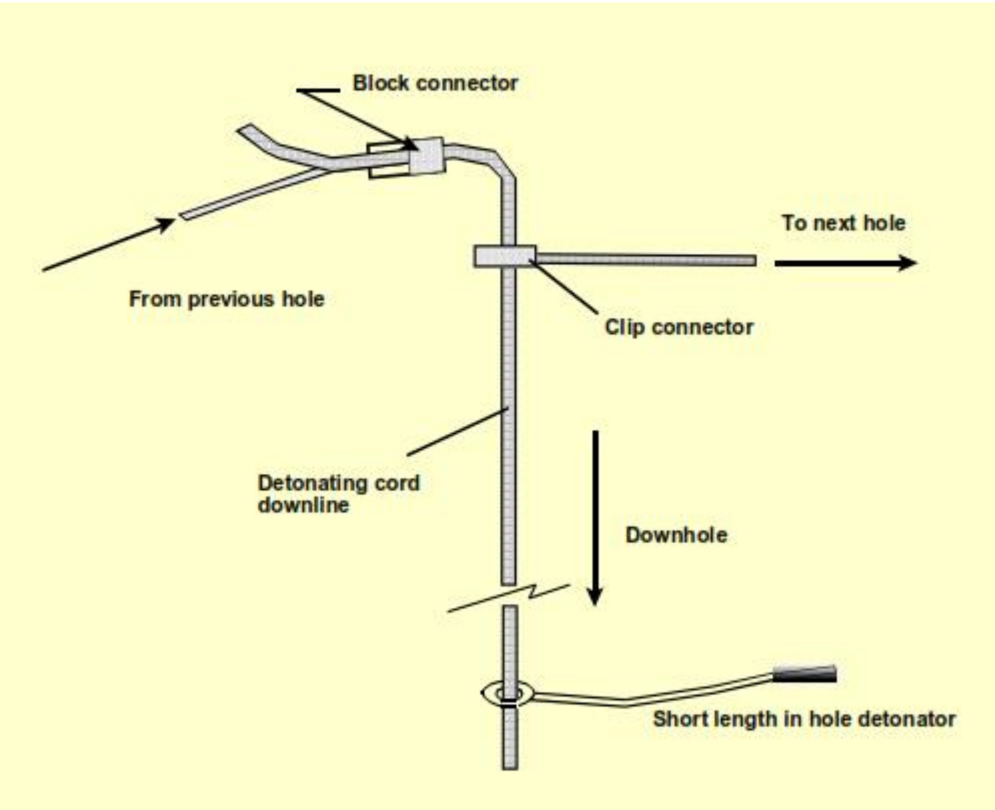

Figure 12 Detonating Cord Set-up

Source: Best D., 2008

Figure 13 shows the Right-Angle Connection applied when detonating cords are used;

Figure 13 Detonating Cords connected at Right Angle

Figure 14 shows an electric initiation device; **Source: figures 13, 14 & 15; Best D., 2008**

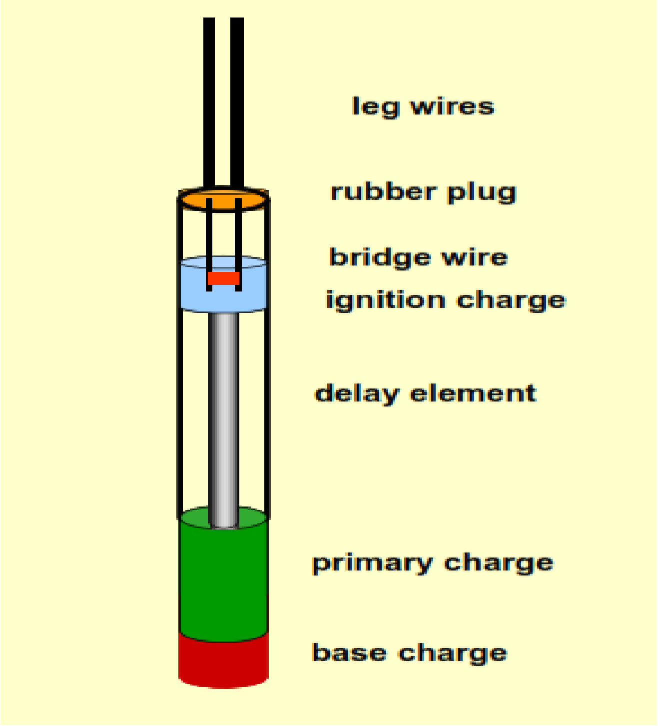

Figure 14 Electric Initiator

The most common and modern initiation systems are electronic initiators. Electronic initiators are designed differently depending on the manufacturer and the engineer should consult the respective manufacturer of the system in order to become acquitted with them.

There are some electronic initiators that are basically designed whereby the electronic detonator is computer-programmed at the factory and can then be initiated just like an ordinary shock tube, however, the electronic detonator contains a timing oscillator and a piezoceramic device.

Very complex electronic initiators also exists and they need a computer at the mine site to fire their detonators, these complex systems are computer-programmed in the field and not at the factory hence they give the engineer more flexibility in terms of altering different input parameters depending on the blasting operation, this is an advantage when the engineer is involved in large scale rock blasting operations but requires the engineer to be very knowledgeable in programming the computer that fires the electronic detonator.

Figure 15 shows the differences between an electronic and electric initiator;

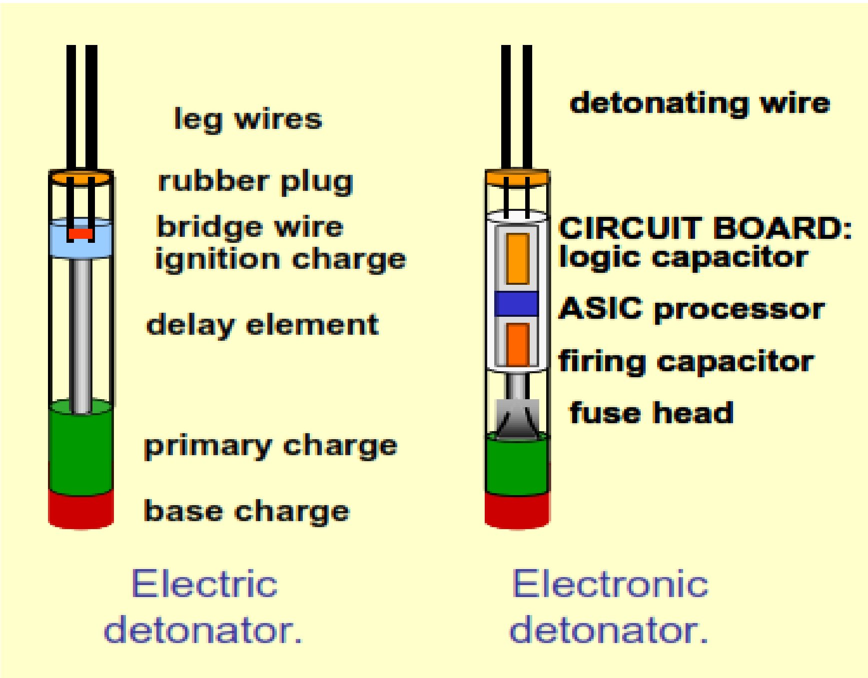

Figure 15 Electronic & Electric Detonators

There are notable differences between Factory-Programmed Initiation Systems and Field-Programmed Initiation Systems and these include;

Field-Programmed Systems;

- No fixed delay times
- Have Electronic Memory for programming before loading or before firing

Factory-Programmed Systems;

- Have fixed delay times
- May include surface connectors for maintaining electrical polarity and improving the tie-in efficiency

Figure 16 Field-Programmed Initiation System

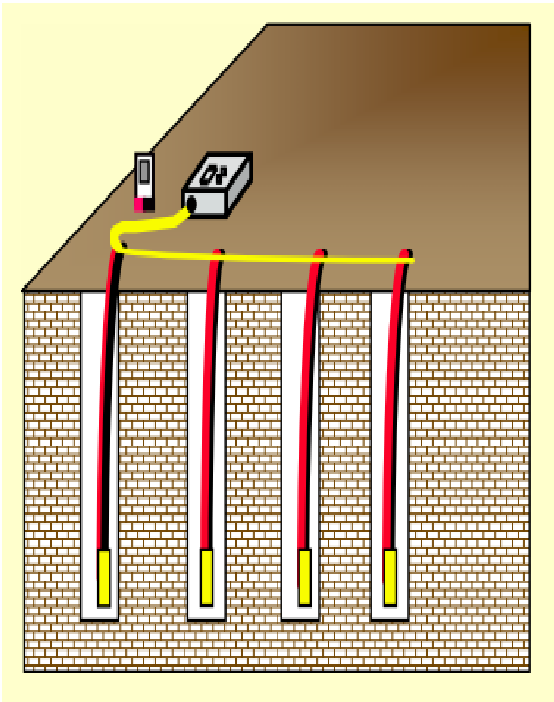

Figure 17 Variable Delay Time in Field-Programmed Systems

Source: Best D., 2008

Through common practice and guidelines from electronic initiation systems manufacturers, engineers are recommended to follow the following practices when working with electronic initiators;

- Engineers should always follow manufacturer's instructions including hook-up procedures
- Test and verify the integrity of the detonator system before initiating the blast
- In case where a blast has been aborted, all personnel should wait for at least 30 minutes before going back to the blast site
- Detonator leads, connectors and coupling devices should be protected at times before the blasting operation
- Wire ends, connectors and fittings should be clean at all times and free from dirt to avoid contamination
- Electronic initiators should always be protected from interference that could damage or affect their performance. Interferences include; electromagnetic fields and radio frequencies
- Electronic detonator wires, connectors and coupling devices should be kept away from any kind of mechanical stress
- Engineers should take extreme care when computer-programming the field electronic initiation systems because wrong programming can cause misfires, excessive fly-rocks, excessive air-blasts and violent vibrations. Programming should take into consideration desired delay times in accordance with the blast design.
- Engineers should never use blasting machines meant for electric detonators on electronic detonators
- Engineers should never mix and use devices from different manufacturers in an electronic initiation system
- Electronic detonators should never be used when there are thunderstorms and lightning, in such cases, all personnel should be evacuated immediately from the mine site
- Engineers should not use electronic detonators outside the specified temperature and pressure ranges

5

ROCK BLASTING AND FRAGMENTATION MECHANICS

5.1 Stages of Rock Fragmentation by Explosives

Rock blasting can be defined as the controlled use of explosives and other methods, for example, plasma processes and gas blasting pressure pyrotechnics, to excavate, break down or remove rock (*Persson, Holmberg R., 1994*).

In this dissertation, it will be *Rock Blasting* using only explosives.

One of the applications of rock blasting in mining is to remove unwanted rock in order to free up the ore body from which minerals can then be extracted.

During rock blasting, the rock fragmentation process follows the detonation of explosives in a drill hole. The explosion itself is a very rapid combustion, where, the energy contained in the explosives is released in form of heat and gas pressure (*Terasvasara, 2006*).

The transition acts on the rock in three consecutive stages (compression, Reflection and Gas Pressure) as shown in **figure 18**;

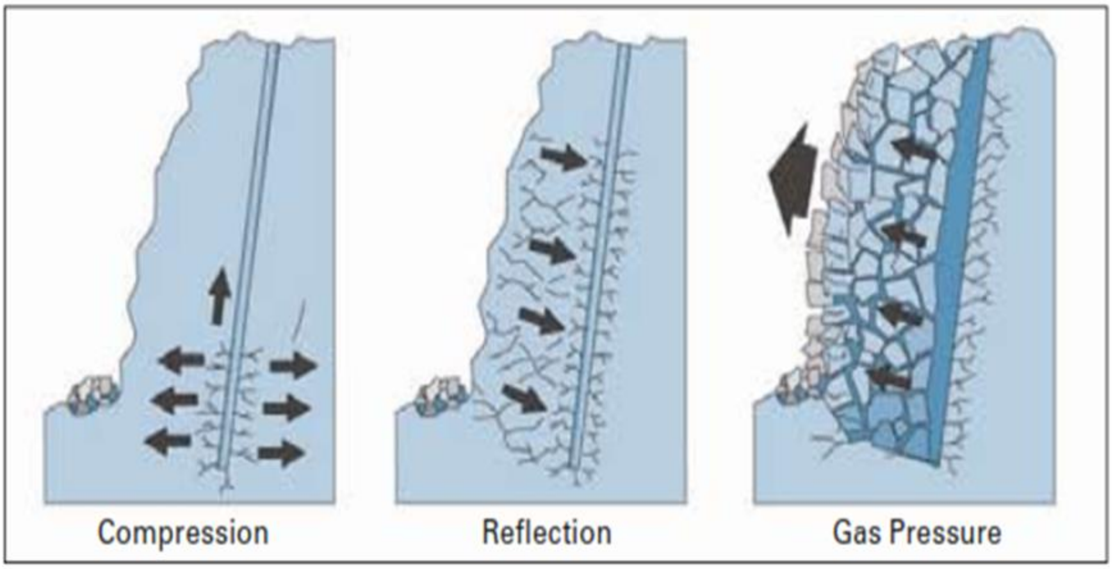

Figure 18 a 3-Stage Rock Breaking Sequence during Normal Blast

Source: Terasvasara, 2006

The three consecutive transition stages are;

a) *Compression*

During the first stage a pressure wave propagates through the rock at a velocity of around 2500 to 6000 m/s, however, the velocity depends on rock type and the type of explosives. In this stage the pressure waves creates micro-fractures which then promote rock fracturing.

b) Reflection

During this next stage the pressure wave bounces back from the free surface, normally from a bench wall or from the natural fissures in the rock itself. At this stage the compression wave is now transformed into shear and tension waves, this transformation increases the rock fracturing process.

c) Gas Pressure

In the third stage, large volumes of gas are released which enter and expand the cracks in the rock under pressure. If the distance between the blast-hole and the free face is calculated correctly the rock mass yields and is thrown forward.

Figure 19 shows a delay detonation of a typical bench blast;

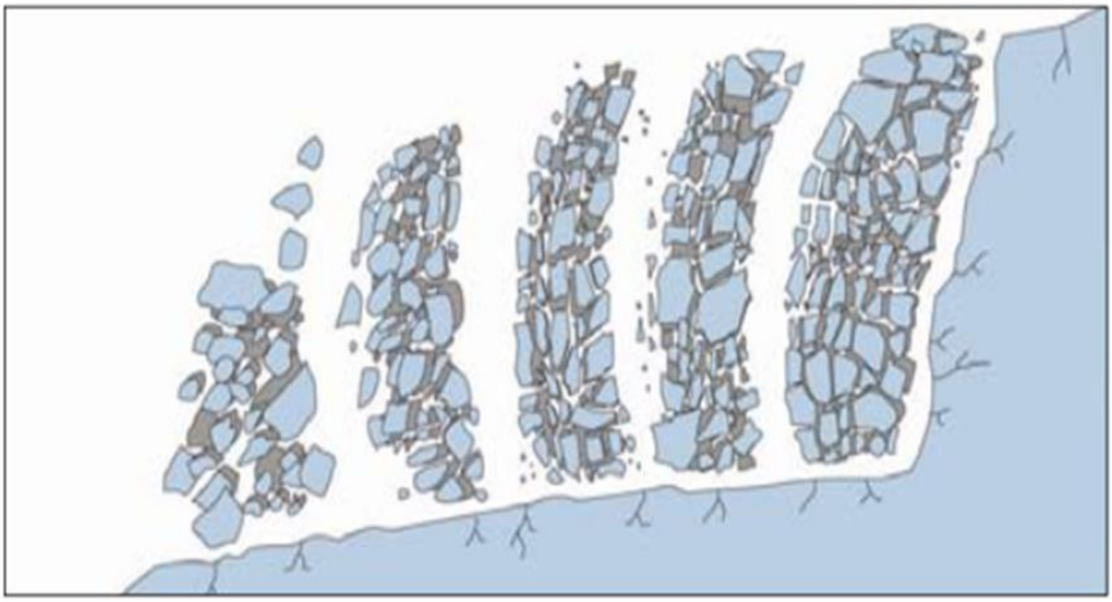

Figure 19 Bench Blasting delay detonation

Source: Terasvasara, 2006

5.2 Rock Blasting Stages

Rock Blasting is grouped into three main stages which include; Pressure Build-up, Wave Transmission and Air-blast.

a) Pressure Build-Up

During this stage the pressure builds up to a peak value and it is associated with shock waves.

Normally explosion gases occupy a much greater volume at ordinary confining pressures than the original explosive charge and are therefore capable of building up transient peak pressures of about 10^5 atm or even more within the vicinity of the charge.

A shock wave is generated within a few milliseconds following detonation and it propagates away from the explosive charge, all rocks within the immediate vicinity are shuttered even the toughest and strongest rocks.

b) Wave Transmission

Work (Force $\times$ Distance) is done in crushing the rock surrounding the charge and eventually the initial shock wave intensity begins to decay after leaving the detonation point.

At a short relative distance the compressive pulse is reduced to an intensity level below the compressive strength of the rock and from this point rock crushing stops, however, the primary (P) and the shear (S) waves continue through the rock mass.

The velocity of the P-wave (v_p) varies depending on the elastic properties of the rock;

$$v_p = \sqrt{\frac{K+\frac{4}{3}\mu}{\rho}} = \sqrt{\frac{\lambda+2\mu}{\rho}} \tag{5.10}$$

v_p – Velocity of P-wave

K – Bulk Modulus (modulus of incompressibility)

μ – Shear Modulus (sometimes given as G (rigidity modulus), also called 2nd Lamé Parameter)

λ – 1st Lamé Parameter

ρ – Density (shows least variation, so v_p is controlled by K and μ.)

S-waves on the other hand do not diverge and hence follow the continuity equation;

$$\nabla.\boldsymbol{u} = 0 \tag{5.11}$$

The velocity of the S-wave (β) can then be characterized as;

$$\beta = \sqrt{\frac{\mu}{\rho}} \tag{5.12}$$

The velocity of P-wave in weak rock travels at approximately 1524 m/s – 3048 m/s and in strong rock it can travel up to 6096 m/s (*Ash R. L., 1963*).

P and S waves carry out *Work* (Force × Distance) by moving the rock particles longitudinally and transversely and because of this the waves attenuate until they consequently die out or encounter a free face.

When engineering damage and vibration control, P and S waves are very important and during construction blasting the distance of travel for the P and S waves is measured in thousands of metres.

c) Air-blast

When a portion of explosive energy reaches the free face as a P-wave it is transferred into the air hence forming an air-blast (*Terasvasara, 2006*). **Figure 4** (**Source: Wikipedia**).

Figure 20 Rock blasting in Honkanummi, Finland

Explosive attack on rock is an extremely violent process and experimental attempts to define the mechanics of rock breakage by explosives to date are still being studied and developed.

When a detonation wave passes through an explosive charge the rock around the blast-hole is subjected to dynamic loading which is then followed by quasi-static loading as follows;

a) dynamic loading, during detonation of the explosive charge, and generation and propagation of the body wave in the medium
b) quasi-static loading, under the residual blast-hole pressure applied by the detonation product gases
c) release of loading, during the period of rock displacement and relaxation of the transient stress field

Figure 21 shows loading that is subjected to the surrounding rock in the vicinity of a blast-hole, a, b, c, = dynamic loading and d = quasi-static loading; Source: **Kutter, Fairhurst, 1971**

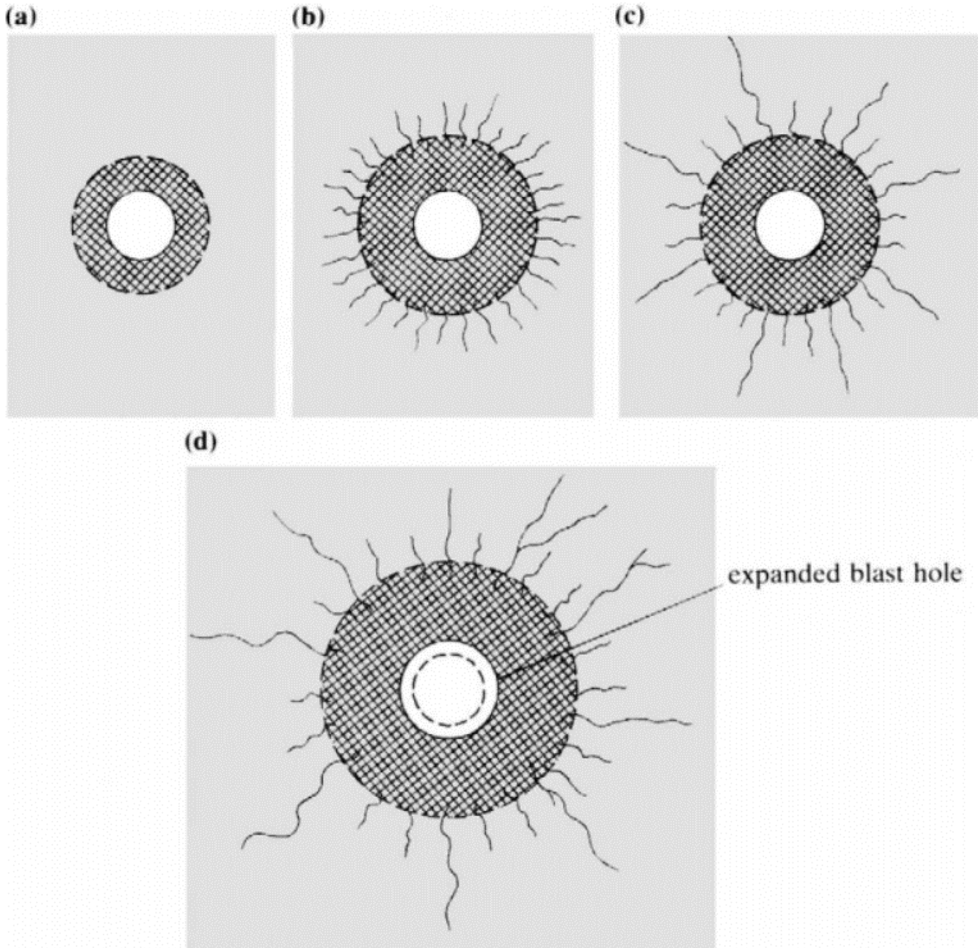

Figure 21 Destruction Stages of rock during blasting under dynamic and quasi-static loading

5.3 Physical Effect Model of Explosives Detonation

*The **Physical Effect Model of Explosive Detonation*** can be applied in open-pit rock blasting in order to estimate the Potential Energy of the muck pile.

The model calculates an energy balance of the blasting operation for a given rock mass.

Cooper in 1996 proposed that the detonation pressure be calculated as;

$$P_0 = \frac{\rho_s c_d^2}{4} \tag{5.13}$$

P_0 – Detonation Pressure (kg/m s^2)

ρ_s – Explosive Density (kg/m^3)

c_d – Detonation Velocity (m/s)

The Energy of the Blasting Charge (E_s) is then calculated using the detonation pressure from equation 5.13 and the total volume of explosive for the whole blasting system (V_s).

$$E_s = V_s \left(\frac{\rho_s c_d^2}{4}\right) \tag{5.14}$$

E_s – Energy of Blasting Charge (kg m^2/s^2)

V_s – Volume of explosives for the whole blasting operation

Through research it has been found that the effective detonation pressure per unit volume (P_{Z0}) is related to the Spacing (λ_s) (borehole distance against the burden), the relationship can be mathematically expressed as follows; (*Muller B., 2010*) ;

$$P_{Z0} = \lambda_s \xi V_{s0} \left(\frac{\frac{\rho_S c_d^2}{4}}{w' \times a'_B \times l_{B0}} \right) \tag{5.15}$$

P_{Z0} – Effective Detonation Pressure of Explosive per unit Volume (Pa = kg/m s^2)

λ_s – Spacing (m)

ξ – Fill Factor (the volume of explosive vs. volume of the blast-hole)

V_{s0} – Volume of explosive per blasted unit volume (m^3)

w' – Average Blasted Burden (m)

a'_B – Average Blasted Blast-hole Distance (m)

l_{B0} – Unit Length of Blast-hole Constant = 1 m

By having the effective detonation pressure per unit volume enables the estimation the Effective Detonation Pressure for the whole system (P_{ZM}) as follows;

$$P_{ZM} = n_{V0} \times P_{Z0} \tag{5.16}$$

Where n_{v0} = the Number of unit volumes.

Because the energy loss during air pressure shock wave propagation, chemical reactions and heat development cannot be quantified, the exhausted energy for vibrations, rock fragmentation and early throw-off phase is calculated as Dynamic Energy (*Muller B., 2010*).

During detonation large amounts of gas fumes are freely released and it is the gas pressure derived from this release that is initially responsible for throw-off of the blasted rock mass and hence the Kinetic Energy of the Thrown-off Muck Pile can be calculated as follows;

$$E_{Kin_M} = \frac{m_M c_M^2}{2} \tag{5.17}$$

$E_{Kin_M} -$ Kinetic Energy of the Muck Pile (J = kg m^2/s^2)

$m_M -$ Muck Pile Mass (kg)

$c_M -$ Blow-out Velocity of Muck Pile (m/s)

The Energy Balance is completed by calculating the **Potential Energy of the Muck Pile**;

$$E_{Pot_M} = \Delta h_w \times g \times m_M \tag{5.18}$$

$g -$ Acceleration due to Gravity (9.81 m/s^2)

$\Delta h_w -$ Bench or Throw Height (m)

Figure 22 summarises the Physical Effect Model of Explosive Detonation in an open-pit mine;

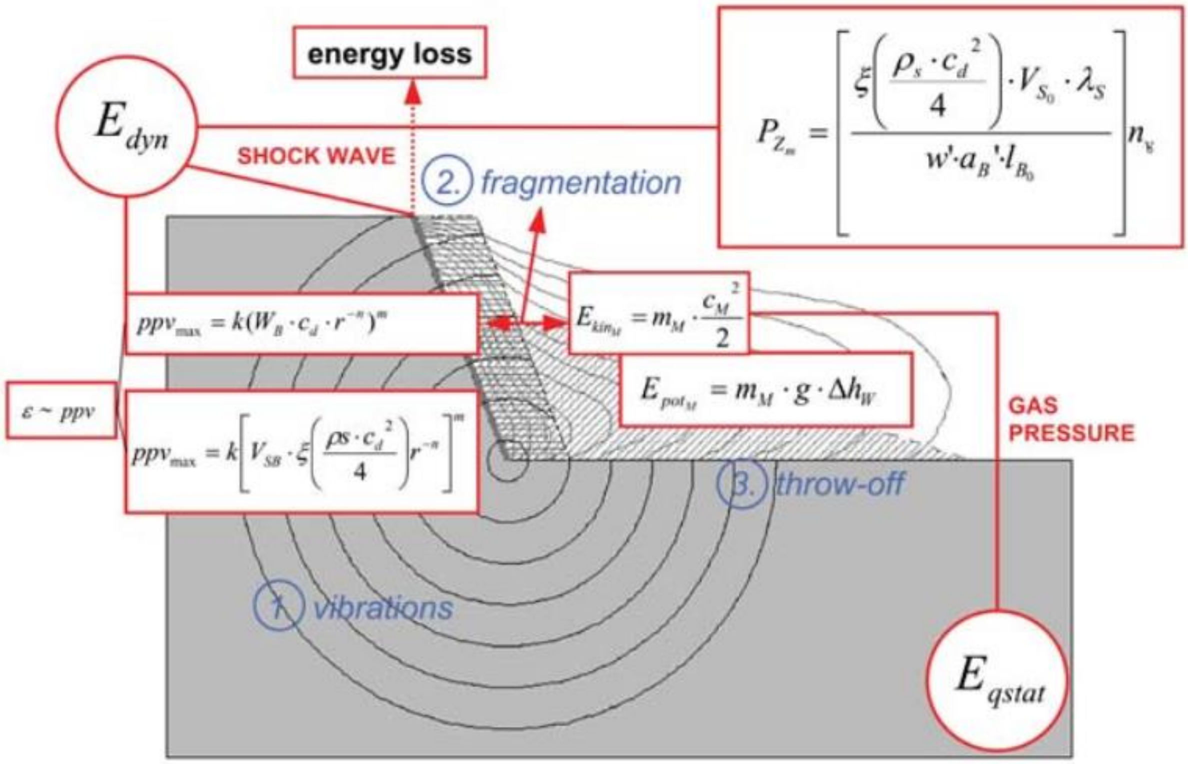

Figure 22 Physical Effect Model used to Calculate Energy Balance during Blasting

Source: Muller B., 2010

E_{dyn} – Dynamic Energy (J = kg m^2/s^2)

E_{qstat} – Quasi Static Energy (J = kg m^2/s^2)

ppv_{max} – Maximum Peak Particle Velocity (m/s)

NB. Quasi-static Thermodynamic Process constitutes the Quasi Static Energy because it happens infinitely slowly and although no real thermodynamic processes are quasi-static, they can be engineered to behave in a quasi-static manner.

At ***Winterberg Quarry*** in ***Germany***, a laser scan was carried during a rock blasting operation in order to show the loosening of the muck pile in the early stages of the fragmentation process, this is shown here in figures 23 and 24, which show a reddish brown part of a bench being loosened during blasting. The blue colour indicates part of the rock that has not yet been fragmented;

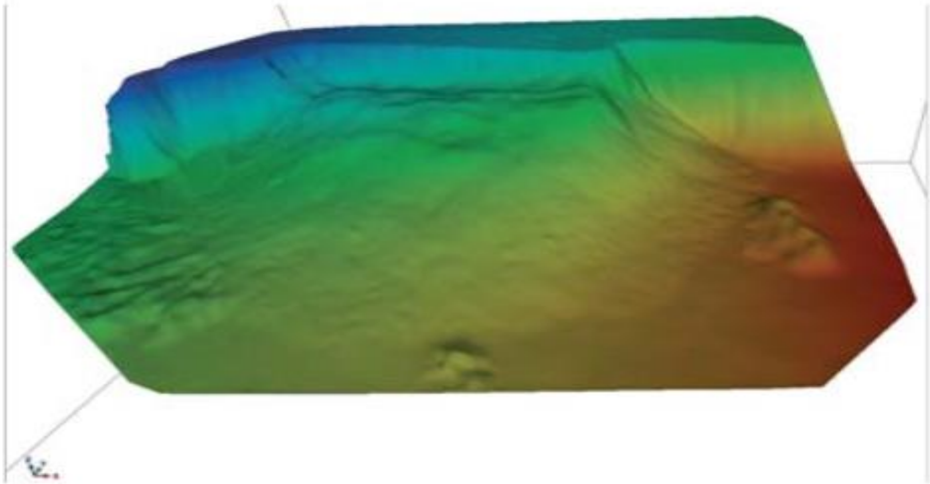

Figure 23 Winterberg Quarry Laser Scan during Blasting

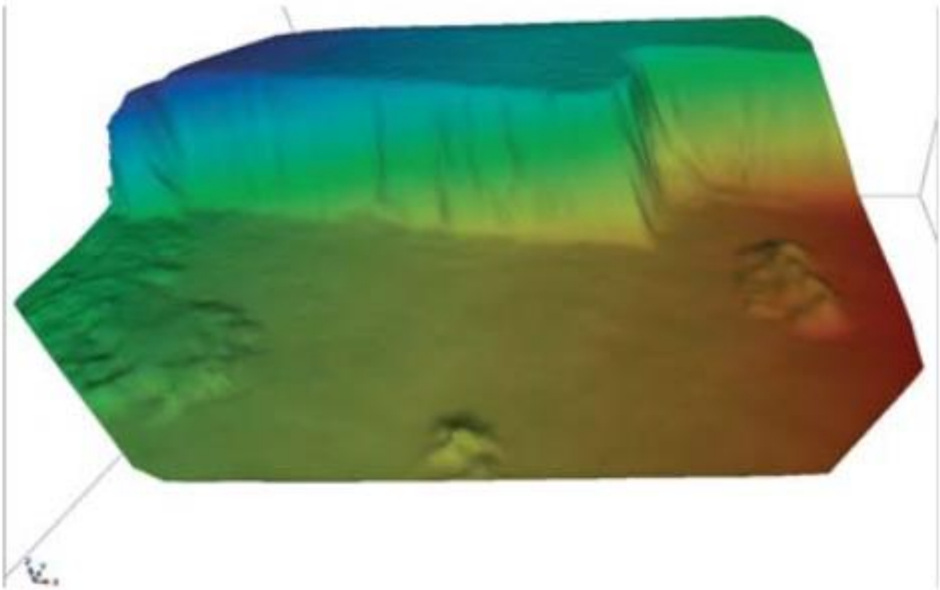

Figure 24 Laser Scan during Blasting at Winterberg Quarry

Source: Muller B., 2010

5.4 Maximum Transmission of Explosive Pressure

When explosives are detonated to break rock there is strain energy which exerts a strong impact on the surrounding rock by the shock wave that is produced, this is followed by the production of Bubble Energy from the explosive gases occurring at very high temperature and pressure.

As discussed in Chapter 3, the Detonation Pressure in **equation 3.19** is,

$$PD = \frac{\rho_e VD^2}{4}$$

Hence the Maximum Pressure Transmitted (PT_m) into the rock can be calculated by;

$$PT_m = \frac{2}{1 + n_z} PD \tag{5.19}$$

Where n_z represents the relationship between the Impedance of the rock and that of the explosive as follows;

$$n_z = \frac{\rho_e \times VD}{\rho_r \times VC} \tag{5.2}$$

ρ_e – Explosive Density (g/cm^3)

ρ_r – Rock Density (g/cm^3)

VD – Detonation Velocity (m/s)

VC – Wave Propagation Velocity through the Rock (m/s)

Equation 5.19 implies that the explosive energy is better transmitted through the rock when the impedance of the rock and the impedance of the explosive are equal making n_z = 1 or something very close to 1 (*Jimeno Carlos L., 1995*).

When $n_z = 1$, from equation 5.19 it means the maximum pressure transmitted through the rock equals the detonation pressure, $PT_m = PD$.

5.5 Induced Ground Vibration

Open-pit rock blasting also results in induced ground vibration and basically a good blast design considers the following factors;

a) Explosive Energy Confinement
b) Explosive Application Experience
c) Explosive Energy Distribution
d) Explosive Energy Level

When explosives are detonated in the blast-hole, there is a chemical reaction between the chemical components of the explosive and this chemical reaction produces a high pressure, high temperature gas. The gas pressure is called *detonation pressure* it is the one that crushes the rock adjacent to the blast-hole, however the detonation pressure decays/dissipates very quickly (*Lucca Frank J., 2003*).

The shock/wave propagation stage follows, after the rock adjacent to the blast-hole is crushed, this is when the wave front moves forward and when it meets a discontinuity some of the energy passes through and some is reflected back. Explosive energy always takes the path of least resistance and once the blasted rock is separated from its bedrock there is no further fracturing because the gas escapes in the empty space created *(Lucca Frank J., 2003*).

In context of explosive rock-blast engineering, the energy that is not utilized in breaking the rock is called *waste energy*, the wasted energy dissipates as vibrations, air-blast and water-shock.

Vibration are wave motions and are created from an energy source, in rock blasting, the source of energy is the explosive energy and rock movement. When the detonating pressure pushes the blasted rock away from the bedrock it results in primary induced ground vibration, the high detonation force causes the rock mass to vibrate and when this vibration is transmitted through the ground, the phenomena is called *propagation*. ***Propagation Velocity*** can hence be defined as the speed at which the vibration wave travels, however as the vibration wave travels away from the initial blast it decays and this is called ***seismic attenuation***.

Good blast design ensures that as much as it is technically possible most of the explosive energy should be used to break the rock, however a poorly designed blast has a lot of vibrations and at higher levels due to wasted energy.

During induced ground vibrations, rock and soil particles also vibrate and the speed at which the rock and soil particles move up and down is called *particle velocity* and the number of times the soil and rock particles move up and down in 1 second is called *frequency*.

The ground particles oscillate in response to the vibration wave and the maximum rate the ground particles can oscillate is called the *Peak Particle Velocity (PPV)*. PPV and Frequency are very important parameters during blast design as they are also specified in legislation to ensure that rock blasting is carried out in a safe manner and to limit the intensity of induced ground vibrations.

PPV is measured in Inches per Second (ips) and the Frequency in Hertz (Hz).

Acceleration is sometimes used in defining the vibration wave peak or intensity, however, it is not recommended to use it as a stand-alone parameter and it should always be looked at in terms of the principal frequency (*Lucca Frank J., 2003*).

Mathematically, the *maximum acceleration* ***(a_m)*** in in/s^2 is;

$$a_m = \frac{2\pi PPVf}{386.4} \tag{5.3}$$

PPV – Peak Particle Velocity (ips)

f – Frequency (Hz)

1 gravity (g) = 386.4 in/s^2

Another important parameter in assessing induced ground vibrations is the *Scaled Distance* (**SD**) which is the scaling factor that relates similar effects of blasting from various charge weights of the same explosive at various distances.

SD is calculated in two ways depending on the situation;

a) Cube Root Scaling;
b) Square Root Scaling

Cube Root Scaling; used for extreme near field tight blasting (under 20 ft. to nearest structure e.g. a house in an urban area) and for the rest Square Root Scaling is used.

In mining, the square root scaled distance is the one commonly applied while the cube root scaled distance is mainly applied in construction blasting where often near-field tight blasting conditions are met.

SQUARE ROOT SCALED DISTANCE

$$SD = \frac{Distance\ to\ Structure}{(Explosive\ Charge\ Weight)^{0.5}} \tag{5.4}$$

CUBE ROOT SCALED DISTANCE

$$SD = \frac{Distance\ to\ Structure}{(Explosive\ Charge\ Weight)^{0.33}} \tag{5.5}$$

For safety reasons only a certain amount of explosives may be detonated per delay, this is called *Delay Sequencing*, for example, an 8 millisecond (ms) delay window may be used. (*Langefors U., Khilstrom B., 1963*).

5.5.1 Quantifying Induced Ground Vibration

In basic terms, the Peak Particle Velocity (PPV) is used to predict intensity of vibrations in terms of ground particle velocity caused by the blasting operation.

The ***Oriard's Formula*** is used as follows;

$$PPV = K\left(\frac{D}{W^{0.5}}\right)^{-1.6} \tag{5.6}$$

PPV – Peak Particle Velocity (ips)

W – Weight of Explosive Charge (lb.)

D – Distance to Nearest Structure (ft.)

K – Confinement Factor with the following values depending on the situation;

K = 20 – Lower Bound

K = 150 – Average

K = 242 – Upper Bound

K = 605 – Highly Confined

Figure 25 shows a Blasting Level Chart of Recommended PPV values with regards to Ground Vibrations caused by rock blasting close to houses from the US MSHA; Seismographs with geophones are used to measure induced ground vibrations from blasting.

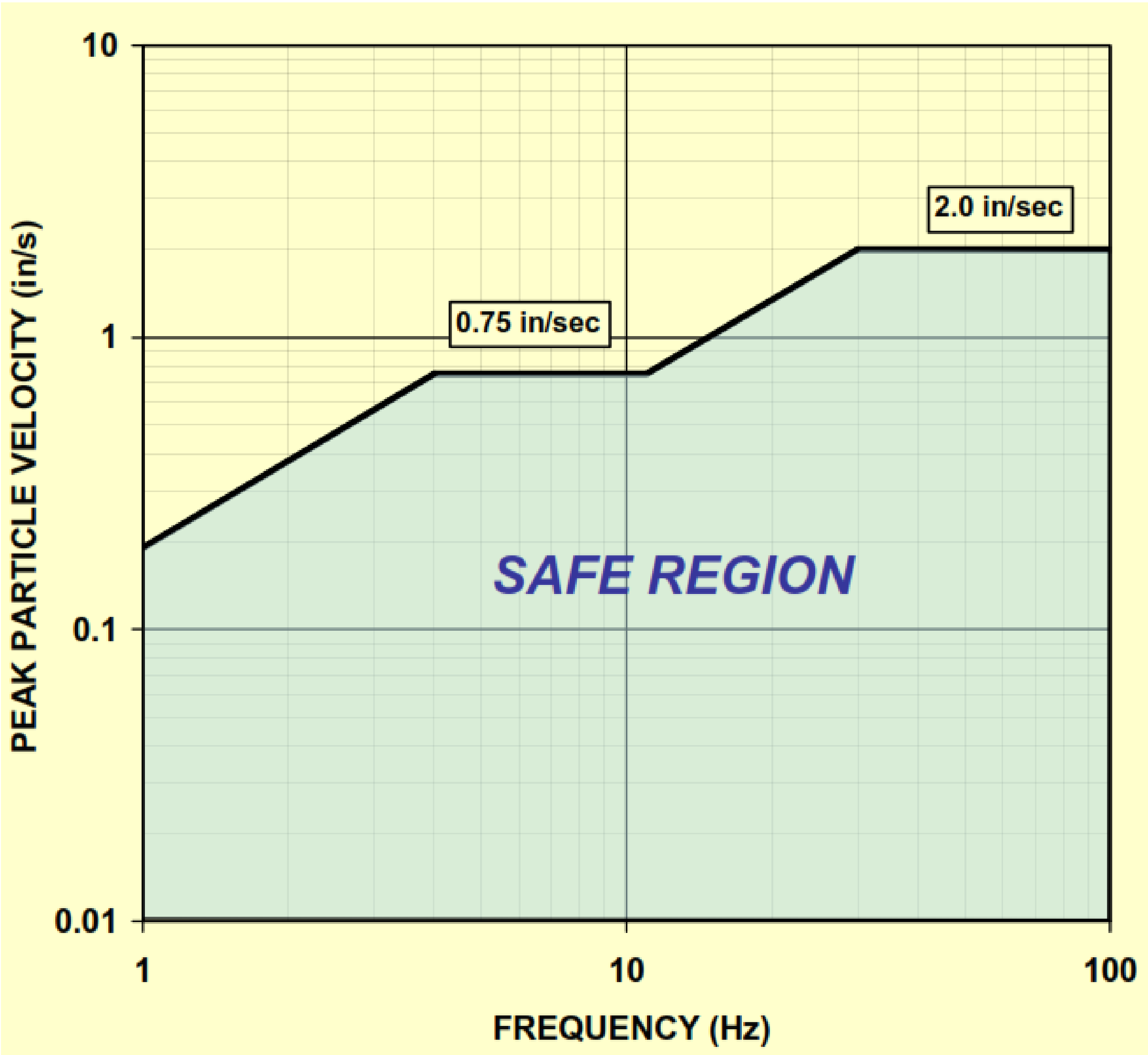

Figure 25 Safe Levels of Blasting Vibrations for Houses

Source: US MSHA, 2014

5.6 Misfires

A Misfire is when there is a complete or partial failure of a charge to explode and hence detonate. Misfires present a very dangerous environment to mine operators because the explosive charges that remain unexploded either in the ground or the muck pile may explode when triggered accidentally by drilling, digging, crushing and any mechanical activity during the mining operation.

In most cases the failed explosive charge and detonator remains either in the face or the muck pile, the danger comes in when, accidentally these charges are detonated by drilling into them, by striking them with site vehicles like excavators; the bucket, the wheels of the vehicle could accidentally hit the failed explosive charge and trigger it to detonate, another case could be when the failed explosive charge after being left in the muck pile is accidentally fed into a crushing plant and the mechanical action of crushing triggers the charge and it explodes.

It is important as far as misfires are concerned for the Mine Manager to take preventive measures rather than trying to manage the situation when misfires have occurred.

Misfires of blast-hole charges when good quality fuses are used are divided into two;

a) Misfires from fuse breakage in the non-active part of the blast-hole
b) Misfires as a result of damage or breakage either in the branch or main line of the fuse

Research has shown that the first group of misfires in blast-holes are mainly caused by the rock shifting more than 15 mm in the non-active part of the blast-hole causing a breakage in the blasting fuse hence ending up in a misfire, the shift of the rock occurs after primary charges are fired and if either the rock is weak or due to insufficient tamping material or both, while the second group of misfires are mainly due to the shockwave from the primary blast or rock shift from the unfired zone both affecting the fuse angle at the junction and in turn making the used fuses come closer to each other, hence, as a result are damaged, this damage leads to a misfire (*Grechkovskii B., 1975*).

5.7 Geomechanics and Rock Blasting

Geomechanics is very significant when carrying out a rock blasting operation because for one it highlights the geomechanical properties of the rock e.g. shear strength, compressive strength, etc. being blasted hence allows the engineer to design the best blast sequence suitable to that particular rock depending on those geomechanical properties.

In order to better understand how the rock will behave geomechanically when blasted, the engineer can use the ***Bieniawski's Rock Mass Rating System*** (RMR) which incorporates ***Rock Quality Designation*** (RQD).

The RQD quantitatively estimates the quality of the rock mass from sample drill core logs, in other words it is the percentage of intact core pieces longer than 4 inches (100 mm) in the total length of the core sample. Core samples should have a diameter of at least 2.15 in. (54.7 mm).

The Bieniawski Rock Mass Classification system uses the following parameters;

a) Rock Quality Designation (RQD)
b) Spacing of Discontinuities
c) Uniaxial Compressive Strength (UCS) of the rock
d) Orientation of Discontinuities
e) Groundwater Conditions
f) Condition of Discontinuities

In mining engineering, there is a slight modification to Bieniawski's RMR because it was primarily developed for use in civil engineering, the slightly modified RMR is called ***Modified Rock Mass Rating System*** (MRMR) and it takes into account the effects of rock blasting, weathering, induced rock stress, in-situ stress and stress variation (rock stress changes) as these parameters are more relevant in mining.

RQD is calculated as follows;

$$RQD = \frac{Sum\ Length\ of\ Core\ Pieces \geq 100\ mm}{Total\ Length\ of\ Core\ Sample} \times 100 \tag{5.7}$$

Figure 26 shows an example of calculating the RQD;

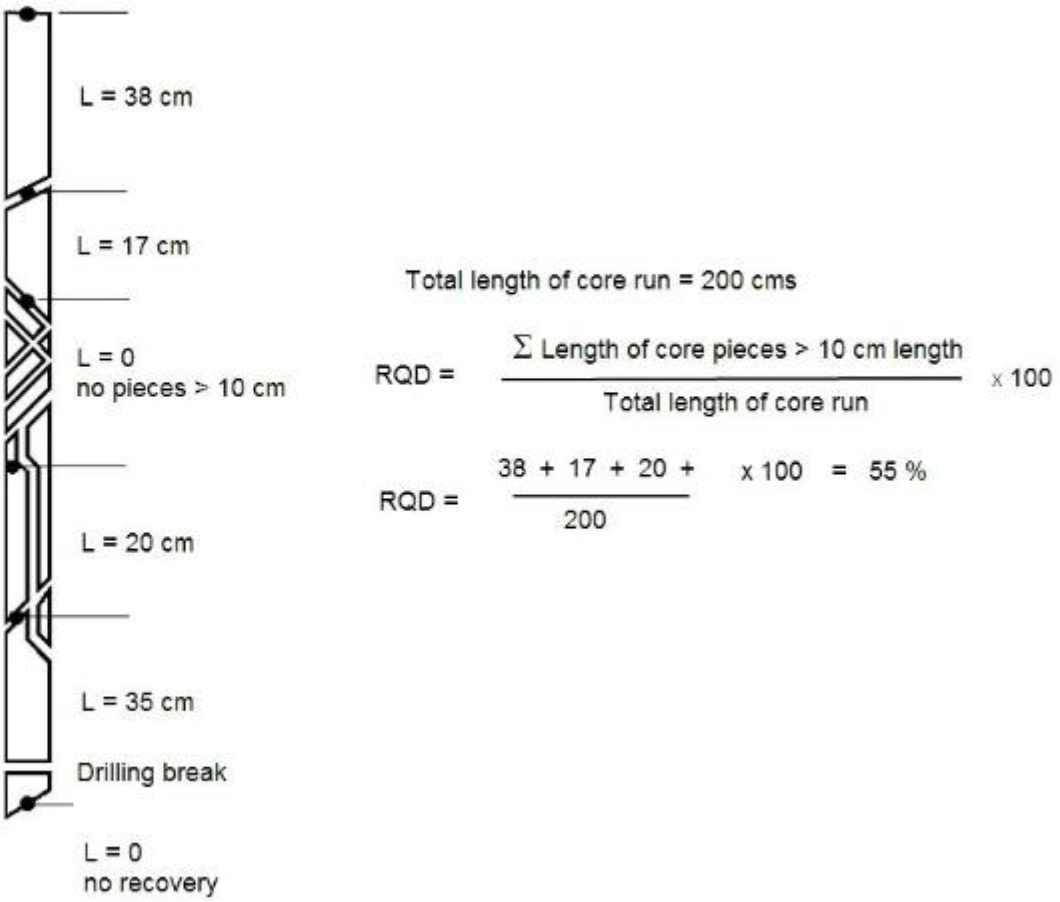

Figure 26 Rock Quality Designation Calculation

Source: Deere, 1989

Table 3 shows Bieniawski's Rock Mass Rating System (RMR); **Source: Bieniawski, 1989**

Table 3 RMR Rock Mass Classification

A. CLASSIFICATION PARAMETERS AND THEIR RATINGS

	Parameter		Range of values						
1	Strength of intact rock material	Point-load strength index	>10 MPa	4 - 10 MPa	2 - 4 MPa	1 - 2 MPa	For this low range - uniaxial compressive test is preferred		
		Uniaxial comp. strength	>250 MPa	100 - 250 MPa	50 - 100 MPa	25 - 50 MPa	5 - 25 MPa	1 - 5 MPa	< 1 MPa
	Rating		15	12	7	4	2	1	0
2	Drill core Quality *RQD*		90% - 100%	75% - 90%	50% - 75%	25% - 50%	< 25%		
	Rating		20	17	13	8	3		
3	Spacing of discontinuities		> 2 m	0.6 - 2 . m	200 - 600 mm	60 - 200 mm	< 60 mm		
	Rating		20	15	10	8	5		
4	Condition of discontinuities (See E)		Very rough surfaces Not continuous No separation Unweathered wall rock	Slightly rough surfaces Separation < 1 mm Slightly weathered walls	Slightly rough surfaces Separation < 1 mm Highly weathered walls	Slickensided surfaces or Gouge < 5 mm thick or Separation 1-5 mm Continuous	Soft gouge >5 mm thick or Separation > 5 mm Continuous		
	Rating		30	25	20	10	0		
5	Groundwater	Inflow per 10 m tunnel length (l/m)	None	< 10	10 - 25	25 - 125	> 125		
		(Joint water press)/ (Major principal σ)	0	< 0.1	0.1, - 0.2	0.2 - 0.5	> 0.5		
		General conditions	Completely dry	Damp	Wet	Dripping	Flowing		
	Rating		15	10	7	4	0		

B. RATING ADJUSTMENT FOR DISCONTINUITY ORIENTATIONS (See F)

Strike and dip orientations		Very favourable	Favourable	Fair	Unfavourable	Very Unfavourable
Ratings	Tunnels & mines	0	-2	-5	-10	-12
	Foundations	0	-2	-7	-15	-25
	Slopes	0	-5	-25	-50	

C. ROCK MASS CLASSES DETERMINED FROM TOTAL RATINGS

Rating	100 ← 81	80 ← 61	60 ← 41	40 ← 21	< 21
Class number	I	II	III	IV	V
Description	Very good rock	Good rock	Fair rock	Poor rock	Very poor rock

D. MEANING OF ROCK CLASSES

Class number	I	II	III	IV	V
Average stand-up time	20 yrs for 15 m span	1 year for 10 m span	1 week for 5 m span	10 hrs for 2.5 m span	30 min for 1 m span
Cohesion of rock mass (kPa)	> 400	300 - 400	200 - 300	100 - 200	< 100
Friction angle of rock mass (deg)	> 45	35 - 45	25 - 35	15 - 25	< 15

E. GUIDELINES FOR CLASSIFICATION OF DISCONTINUITY conditions

Discontinuity length (persistence)	< 1 m	1 - 3 m	3 - 10 m	10 - 20 m	> 20 m
Rating	6	4	2	1	0
Separation (aperture)	None	< 0.1 mm	0.1 - 1.0 mm	1 - 5 mm	> 5 mm
Rating	6	5	4	1	0
Roughness	Very rough	Rough	Slightly rough	Smooth	Slickensided
Rating	6	5	3	1	0
Infilling (gouge)	None	Hard filling < 5 mm	Hard filling > 5 mm	Soft filling < 5 mm	Soft filling > 5 mm
Rating	6	4	2	2	0
Weathering	Unweathered	Slightly weathered	Moderately weathered	Highly weathered	Decomposed
Ratings	6	5	3	1	0

F. EFFECT OF DISCONTINUITY STRIKE AND DIP ORIENTATION IN TUNNELLING**

Strike perpendicular to tunnel axis		Strike parallel to tunnel axis	
Drive with dip - Dip 45 - 90°	Drive with dip - Dip 20 - 45°	Dip 45 - 90°	Dip 20 - 45°
Very favourable	Favourable	Very unfavourable	Fair
Drive against dip - Dip 45-90°	Drive against dip - Dip 20-45°	Dip 0-20 - Irrespective of strike°	
Fair	Unfavourable	Fair	

* Some conditions are mutually exclusive . For example, if infilling is present, the roughness of the surface will be overshadowed by the influence of the gouge. In such cases use A.4 directly.

** Modified after Wickham et al (1972).

6

BLAST DESIGN

6.1 Introduction

Proper blast design is the most important tool to prevent blasting problems including fly-rock. A blast engineer optimises the balance between rock properties, explosive energy distribution, and explosive energy confinement. The most logical approach is to adjust energy distribution and confinement suitable for the rock properties, including geological abnormality. Such optimization would improve fragmentation and minimize blasting problems including; fly-rock, ground vibration, and air-blast.

Today, with the expanded use of computers and electronic aids there are more tools to assist the blast engineer in designing a safe and efficient blast. Burden, spacing, hole-diameter, stemming, sub-drilling, initiation system, and type of explosive used should match the characteristics of the rock formation. For example, a closely jointed blocky limestone formation would require a tighter pattern (such as 243cm by 243cm, or less) with small diameter blast-holes than a competent homogeneous sandstone formation (*Lobb T.E., 2002*).

In open-pit mining just like in underground mining there are guidelines which are applied during blast design, the guidelines provide an initial estimate from which to design a blasting pattern, however, it is also very crucial and vital to consider the properties of the rock and the properties of the explosive during blast design. The guidelines for blast design in open-pits have been established for years and represent common practice in open-pit mining engineering.

Below are images which illustrate aspects of blast design in open-pit mines and quarries;

Figure 27 shows common nomenclature used for bench blast design in open-pit mining;

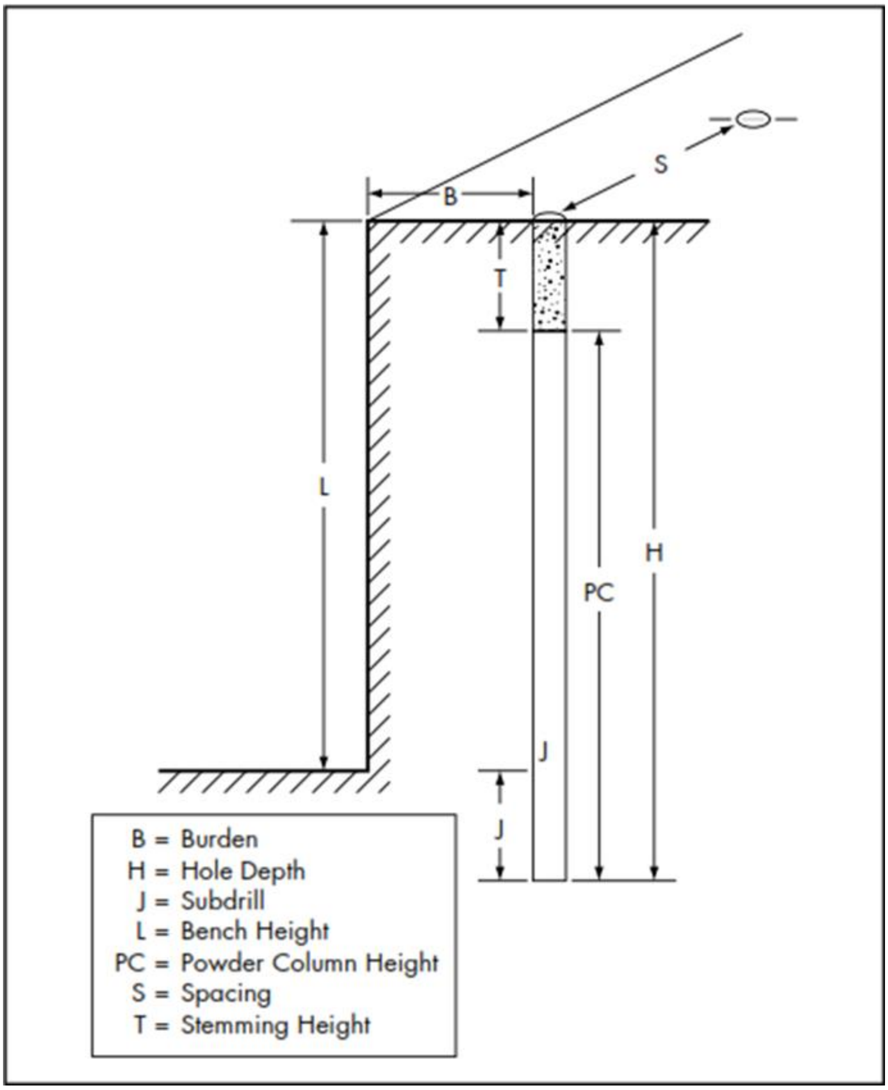

Figure 27 Nomenclature for Bench Blast Design

Source: Darling Peter, 2011

Figure 28 elaborates further common terminology used in Blast Design;

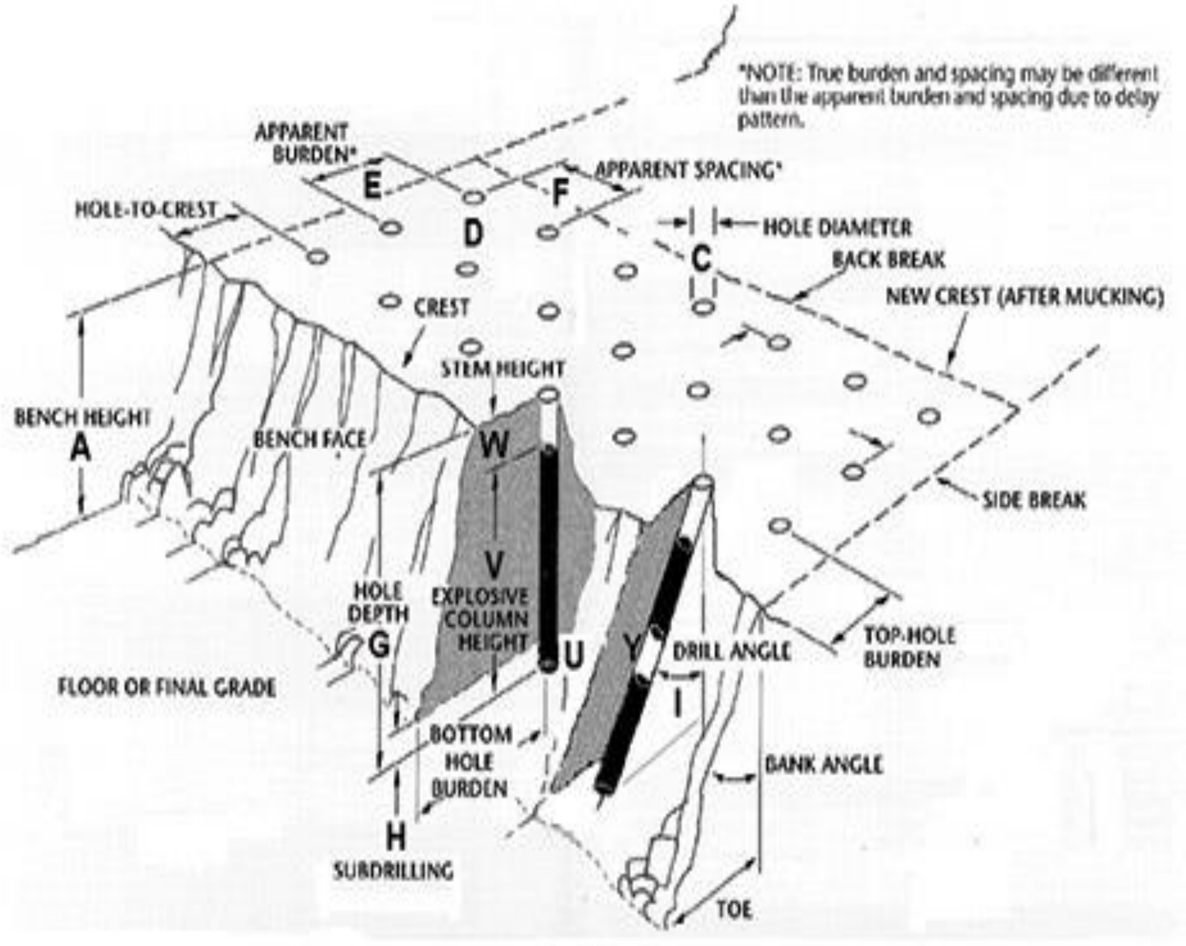

Figure 28 Common Terminology in Blast Design

Source: GeoDrilling International, Web Image, 2014

Figure 29 shows a 2D view of bottom charging and column charging;

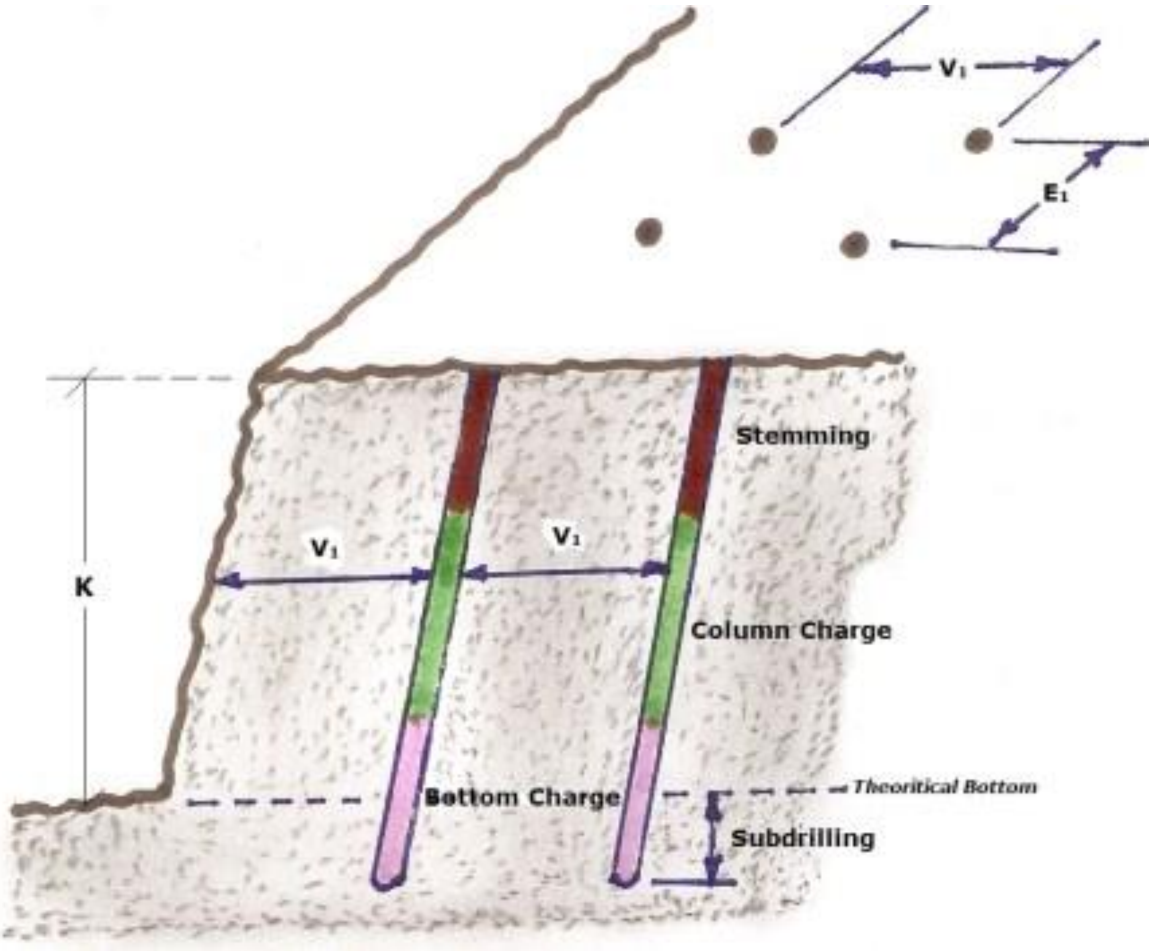

Figure 29 Column and Bottom Charging

Source: GeoDrilling International, Web Image, 2014

Figure 30 further illustrates blast design terminology in open-pit mines and quarries;

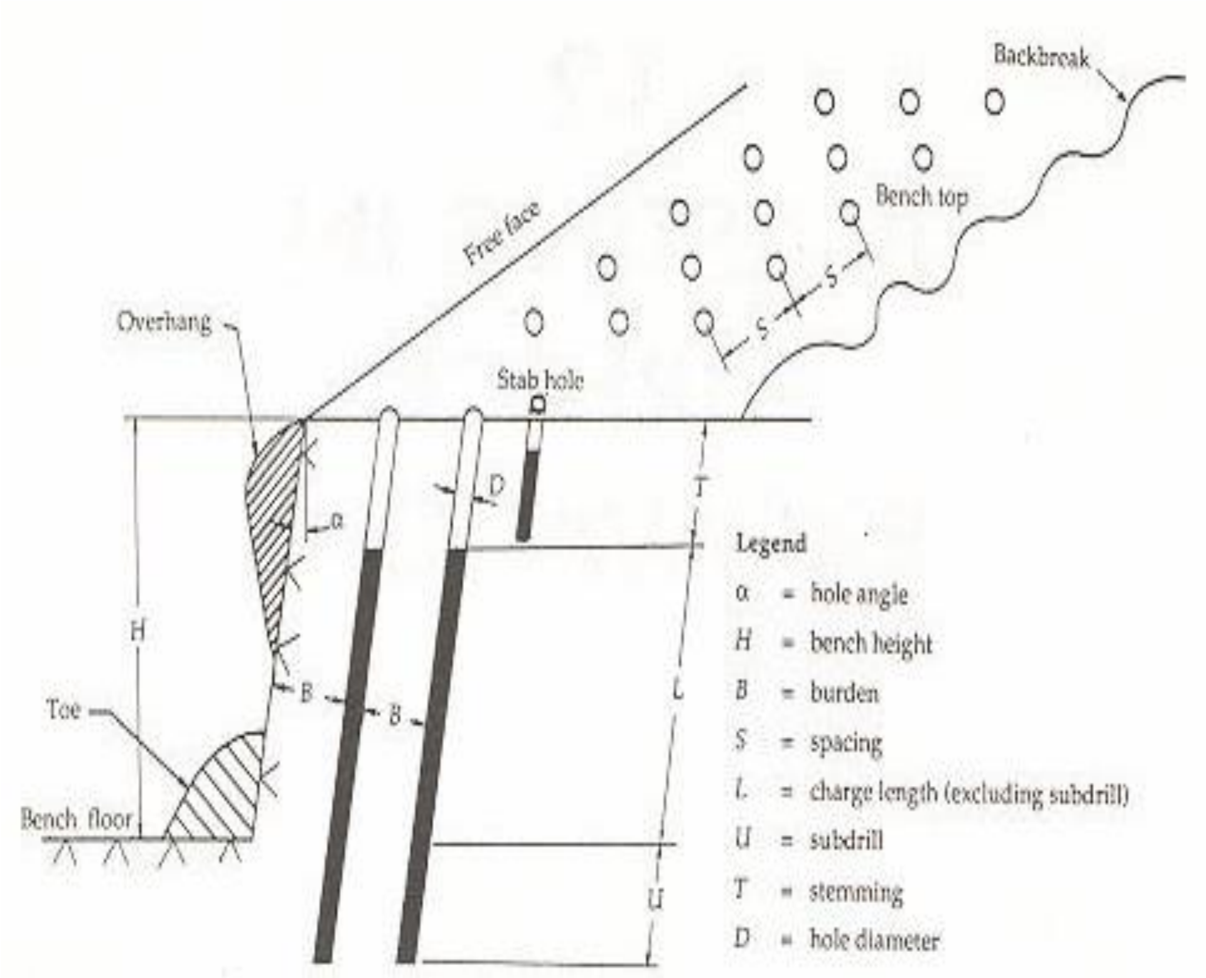

Figure 30 Blast Design Terminology

Source: GeoDrilling International, Web Image, 2014

Figure 31 shows different initiation patterns for firing explosives shots in open faces, the patterns include; Square V, Square V1, Square V2, Staggered V, Staggered V1and Staggered V2.

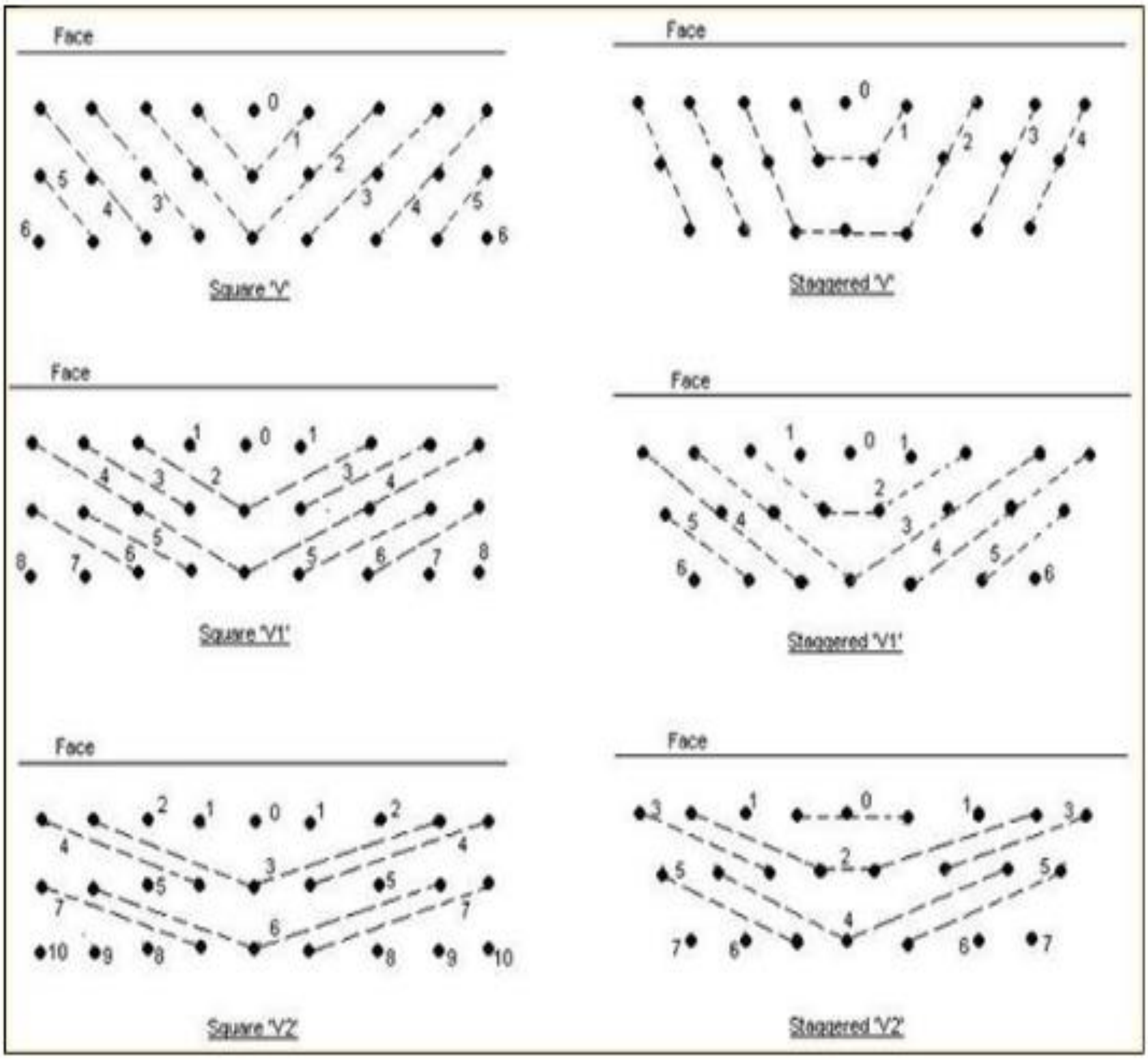

Figure 31 Different Initiation Patterns for Shot Firing in an Open Face

Source: WordPress Web Image, 2014

6.2 Characteristics of a Good Rock Fragmentation

Rock fragmentation is an important part of the mining operation (Singh S.P., 2013) and basically depends on two variables; Rock Mass Properties – that cannot be controlled and Blast Design Parameters – which can be controlled.

By optimising blast design parameters, costs of downstream operations can be reduced because the target rock fragmentation goal is met thereby removing the need for further fragmentation.

In every mining operation there are two vital aspects which influence how the mining is carried out, these are; operation optimisation and cost reduction i.e. the mining operation should be carried out with the highest efficiency possible at a reduced cost that maximises profits for the mining company without incurring financial loss.

Rock fragmentation by blasting produces a fragmented rock, and if the blast design parameters are well executed during rock blasting, the operation produces rock fragments which have desirable characteristics, in other words, by looking at the fragmented rock engineers can tell if the rock fragmentation operation was good or not good, that is to say; whether rock fragmentation produced what it was intended to produce.

In mining, no single fragment size distribution satisfies every need. For example, in some operations, fine fragmentation is preferable because the rock is eventually going to be ground down to powder, as with copper ore or limestone (for cement production). However, in iron ore mining and the quarrying of aggregates, fines (screenings) are usually a less valuable by-product. Also, in large mining operations, larger equipment such as crushers are used, so they can generally handle larger rock than smaller operations.

Desirable characteristics of fragmented rock that eventually goes to grinding to get reduced to powder size for further mineral processing according to a study done by Singh S.P. in 2004 include;

a) **Mean Fragment Size**

It is desirable from a mineral processing point of view to have smaller rock fragments because this increases the gain in productivity during grinding.

b) **Index of Uniformity (N)**

The Index of Uniformity (N) should be low around 1.1 and 1.3, research done by Chung et al. (1991) concluded that when fragmented material has high N values, the material then has a distribution of fragment sizes which interlock and this results in a tight fragment pile that has a result a higher resistance against penetration by a loader's bucket.

c) **Minimum % of Oversize**

The minimum % of oversize is a very vital desirable characteristic of fragmented rock to look for although it is very difficult to quantify. It is important that the size of the rock fragments produced by rock blasting not to only fit into the loading machine's bucket but to do so without increasing the dig cycle time. When the % of oversize is high, it affects productivity of downstream operations by increasing the need for secondary blasting and increasing the maintenance requirement of machinery thereby increasing equipment maintenance costs (*Abdul H., 2013*).

d) **Reasonable % of Fines**

The reasonable % of fines depends on loading practice and the nature of the rock material. Fine material acts as a lubricant between the coarser fragmented material and this facilitates penetration of the bucket of the loading machinery.

e) **Low Water Content**

When the content of water in the fragmented rock is high, it affects the bulk density of the material and the machine traction thereby reducing the loading efficiency of the loader.

f) **Low Stickiness of Fragmented Material**

It is also desirable to have fragmented material that does not stick, however, this property greatly depends on the nature of the rock, and for example, soft and argillaceous rocks under high saturation levels tend to stick. Sticking fragmented material tends to lower the loading efficiency of a loading machinery.

6.3 Calculating the Explosive Charge

Calculating the charge is another important aspect and part of blast design, the engineering principles used in the charge calculation below have been adopted from the book; *Swedish Blasting Technique (1981) by R. Gustafsson* and today the same principles apply and represent common practice within the mining industry.

<u>Nomenclature for Charge Calculation</u>

B – Burden (it is the shortest distance (m) between the bottom of the drill-hole and free face)

B_{max} – Maximum Burden (m)

B_p – Practical Burden (m)

S_p – Practical Hole Spacing (m)

D – Hole Diameter (m).

L – Bench Height (m)

H – Hole Depth (m)

J – Sub-drilling (m)

F – Faulty Drilling Heave Factor

C_{QP} – Concentration of Column Charge

C_{QB} – Concentration of Bottom Charge

Q_b – Bottom Charge (kg).

Q_P – Column Charge (kg).

h_B – Height of Bottom Charge, h_P – Height of Column Charge

Q – Total Charge

q – Specific Charge (kg/m^3) (amount of explosives consumed per cubic meter of rock)

b – Specific Drilling (drilled metres/m^3) (drilled meters per cubic metres of excavated rock)

Round – refers to a one stage blasting which produces a volume of rock in m^3

C – Rock Blastability (also known as Blasting Ratio or Blasting Coefficient) (kg/m^3)

- It is the charge required by the weight to break 1 m^3 of rock.
- = 0.2 kg/m^3 (easily blasted rock)
- = 1 kg/m^3 (difficult blasted rock)
- = 0.4 kg/m^3 (standard Blasting Ratio)

Formulae for Charge Calculation

$Q = Q_b + Q_p$

$B_{max} = 45D$

$J = 0.3B_{max}$

$H = L + J + 0.05(J + L)$

- 0.05 is 5 cm/m drill with hole inclination of 3:1

$F = 0.05 + 0.03H$

- 0.05 is 5cm/m application error and 0.03 is 3cm/m drill hole

$B_p = B_{max} - F$

$S_p = 1.25B_p$

$C_{QB} = \frac{D^2}{10^3}$

$Q_b = h_B C_{QB}, Q_P = h_p C_{QP}$

$h_B = 1.3B_{max}$

$C_{QP} = 0.5C_{QB}$

$h_P = H - (h_B + h_0)$

$h_0 -$ Height of uncharged section, $h_0 = B_p$ and sometimes taken as $h_0 = B_{max}$

An Example Calculation

Given the following data;

- Bench Height (L) = 10 m,
- Round Width (B) = 26 m,
- Drill-hole diameter (D) = 64 mm,

An engineer is able to calculate the Blast Design parameters for charging as follows;

- $B_{max} = 45 \times D = 45 \times 64 = 2880\ cm = 2.88\ m$
- $J = 0.3B_{max} = 0.3 \times 2.88 = 0.864\ m\ \cong 0.9\ m$
- $H = L + J + 0.05(J + L) = 10 + 0.9 + 0.05(0.9 + 10) = \mathbf{11.4\ m}$
- $F = 0.05 + 0.03H = 0.05 + (0.03 \times 11.4) = 0.39$
- $B_p = B_{max} - F = 2.88 - 0.39 = \mathbf{2.5\ m}$
- $S_p = 1.25B_p = 1.25 \times 2.5 = 3.1\ m$
- $Number\ of\ Hole\ Spacings = \frac{B}{S_p} = \frac{26}{3.1} \cong 8$
- $Corrected\ Practical\ Hole\ Spacing = \frac{B}{8} = \frac{26}{8} = \mathbf{3.25\ m}$
- $C_{QB} = \frac{D^2}{10^3} = \frac{64^2}{10^3} = 4.1$ kg/m
- $h_B = 1.3B_{max} = 1.3 \times 2.88 = 3.7\ m$
- $Q_b = h_B C_{QB} = 3.7 \times 4.1 = \mathbf{15.2\ kg}$
- $C_{QP} = 0.5C_{QB} = 0.5 \times 4.1 = 2.05\ \cong$ **2 kg/m**
- $h_0 = B_p = 2.5\ m$

- $h_P = H - (h_B + h_0) = 11.4 - (3.7 + 2.5) = 5.2\ m$
- $Q_P = h_p C_{QP} = 5.2 \times 2 = \mathbf{10.4\ kg}$
- $Total\ Charge;\ Q = Q_b + Q_p = 15.2 + 10.4 = 25.6\ kg$
- $q = [(\text{holes/row}) \times Q_{Total}] / (B_p LB) = \frac{[8 \times 25.6]}{2.5 \times 10 \times 26} = 0.32\ \text{kg/m}^3$
- $b = [(\text{holes/row}) \times H] / (B_p LB) = \frac{[8 \times 11.4]}{2.5 \times 10 \times 26} = 0.14\ drilled\ metres/\text{m}^3$

Hence Summary of Important Blast Design Charge Parameters from the example calculation are as follows;

Bench Height **(L)** **m**	Hole Depth **(H)** **m**	Practical Hole Spacing **(S_p)** **m**	Practical Burden **(B_p)** **m**	Conc. of Bottom Charge **(C_{QB})** **kg/m**	Bottom Charge **(Q_b)** **kg**	Column Charge **(Q_p)** **kg**	Concentration of Column Charge **(C_{QP})** **kg/m**	Specific Charge **(q)** **kg/m³**	Specific Drilling (b) **drilled metres /m³**
10	11.4	3.1	2.5	4.1	15.2	10.4	2	0.32	0.14

6.4 Elements of Blast Design in Open-Pit Mines

I. The Importance of Free Faces

During blasting, a free face provides direction for movement and hence a degree of freedom. As the number and extent of free faces located near the blast-hole increases, the amount of rock that can be fragmented from that blast-hole also increases, this is shown in **figure 32**;

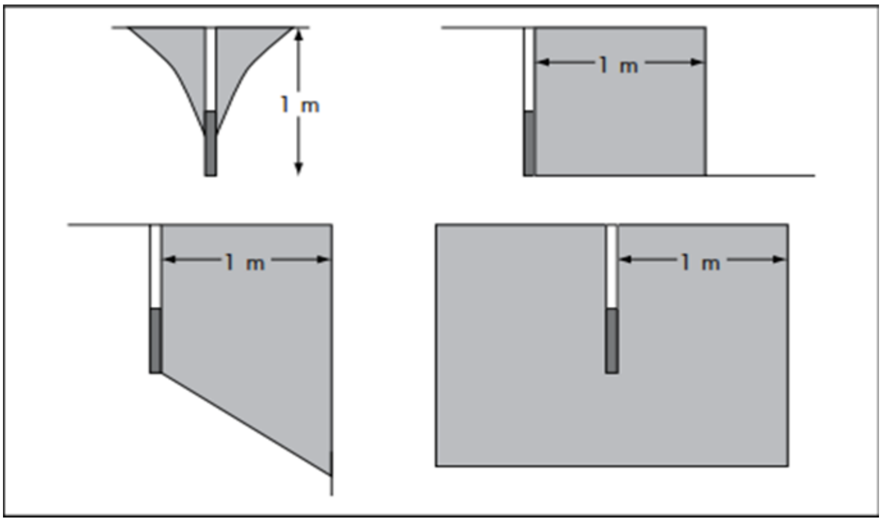

Figure 32 Effect of Increasing Degrees of Freedom on Fragmented Volume with identical holes

Source: Darling Peter, 2011

Hence, because of this geometric factor including properties of the blasted rock and rock mass discontinuities, a proper blast design, thereby, requires more than simply adopting a suitable ***powder factor***. The aim of good blast design is to produce the desired fragmentation with the minimum of back-break and environmental effects like; fly-rocks, high air-blast and induced ground vibrations. When there is excessive fly-rock, air-blast and induced ground vibration they all indicate inefficient use of explosive energy. In order to get coarser rock less explosives are used which implies expanding the drill pattern and to get finer rock more explosives are used with a tighter pattern (*Darling P., 2011*).

The ***Powder Factor*** is the amount of explosives used per unit volume or weight of rock (kg/m^3 or kg/t) blasted and by using blast design equations and guidelines engineers arrive at a reasonable rock fragmentation rather than optimum fragmentation and although the blast design equations are derived from different theories and empirical tests but they do all yield the same powder factor results. **Figure 33** shows powder factors for different mining and construction operations;

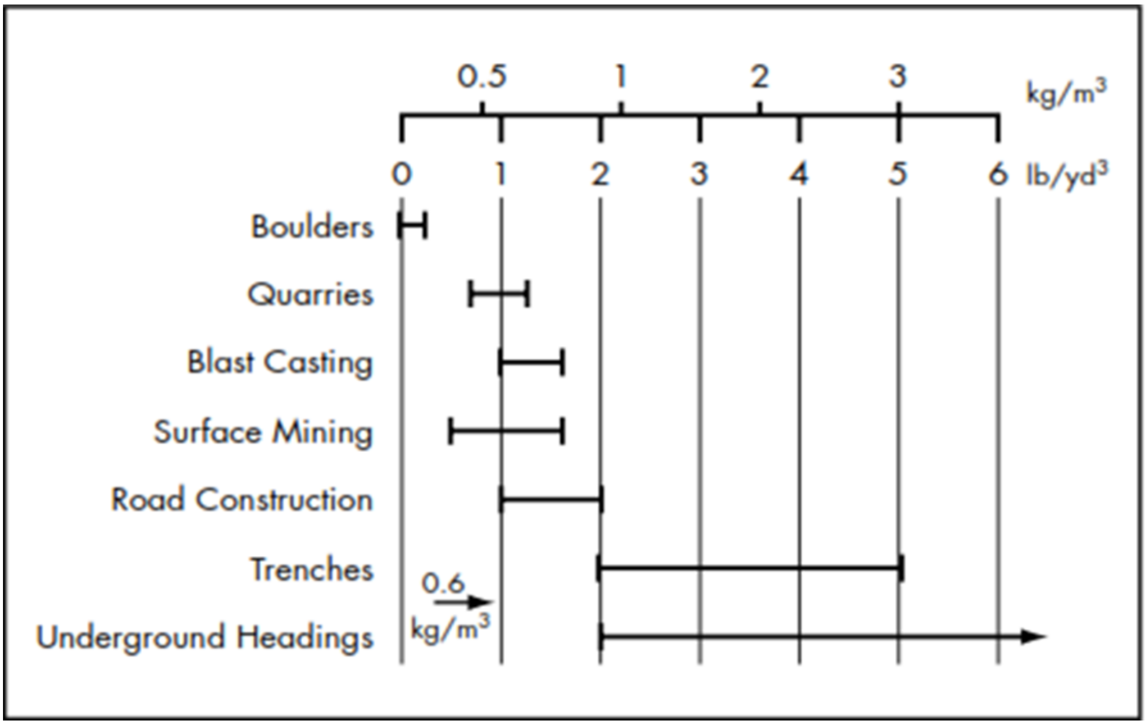

Figure 33 Powder Factor Ranges for different Types of Blasting

Source: Darling Peter, 2011

II. Hole Diameter, Burden (B) against Face Height (L)

This is a very important parameter that is most often overlooked, it is recommended that the face height to burden ratio should be at least 3:1 to 4:1 in order to have a good fragmentation.

When the faces are short, they are stiff and resist breakage and also the amount of stemming which is required to seal the hole increases as the face shortens and this leaves less room for the explosives placed in the hole, thereby, increasing the proportion of the bench interval away from the adjacent explosives columns causing the hole utilisation and powder factors to drop as illustrated in ***figure 34***;

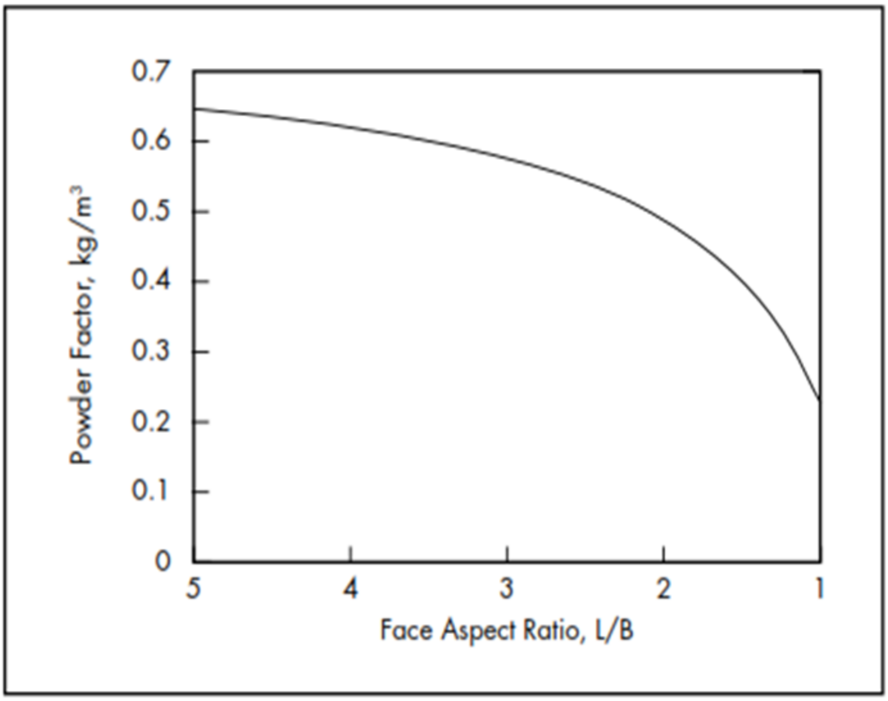

Figure 34 Effect of Face Height on Powder Factor with constant Burden and Spacing

Source: Darling Peter, 2011

As the face height decreases since the stemming height is proportional to the hole diameter and burden, the powder column height decreases disproportionally hence reducing bore-hole utilisation in the explosive filled portion (*Lusk B., 2011*).

So, in order to maintain the powder factor, the pattern must be shrunk and this requires drilling more holes, however, this increases drilling costs, this is shown in ***figure 35*** which elaborates how much drilling is needed to maintain the powder factor;

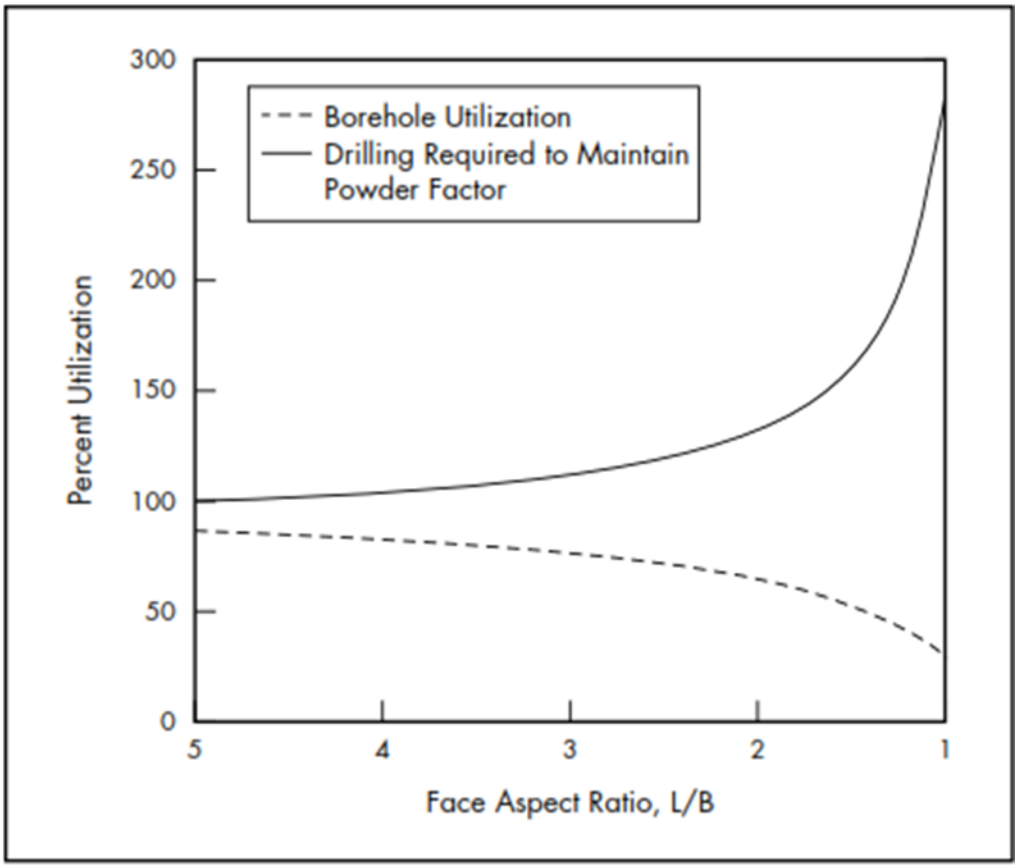

Figure 35 Effect of Face Aspect Ratio on Blast-hole Utilisation

Source: Darling Peter, 2011

In many metal mines it is the ore-body configurations that dictate smaller bench heights for ore selection but it is also the same smaller bench heights that are often used for waste removal (*Lusk B., 2011*).

Figure 36 shows hole utilisation curves for shots designed using open-pit mining common practice, the upper curve represents holes with sub-drilling in metal mining and the lower curve represents holes without sub-drilling in limestone quarrying;

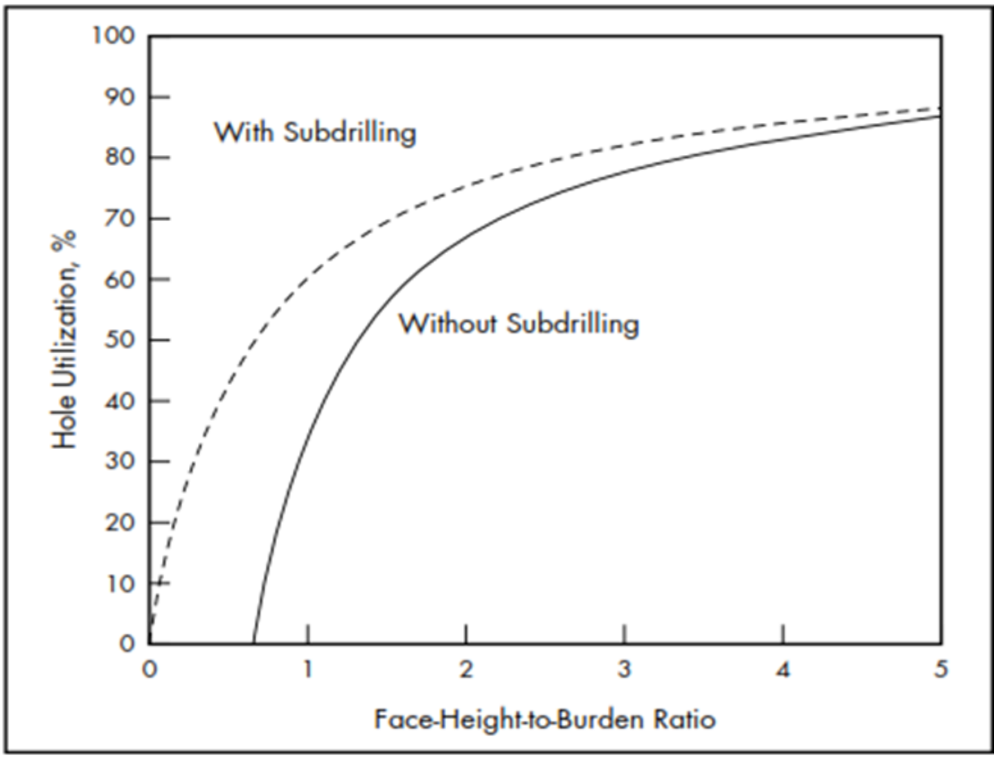

Figure 36 Hole Utilisation Curves with and without Sub-Drilling in Open-Pit Mining

Source: Darling Peter, 2011

When the face height is excessive there is ***drill deviation***; which is the variation in the distribution of explosive energy at the bottom of the blast-holes and drill deviation causes inadequate breakage in areas where the holes wander excessively way from each other and it most often results into high or hard toe. In contrast, when the blast-holes are very close to the face there is most often excessive fly-rock and air-blast.

Figure 37 shows the relationship between drill deviation and face aspect ratio (L/B) such that;

$$Face\ Aspect\ Ratio = \frac{Bench\ Height}{Burden} \tag{5.0}$$

Hence;

$$Face\ Aspect\ Ratio = \frac{L}{B}$$

L – Bench Height

B – Burden

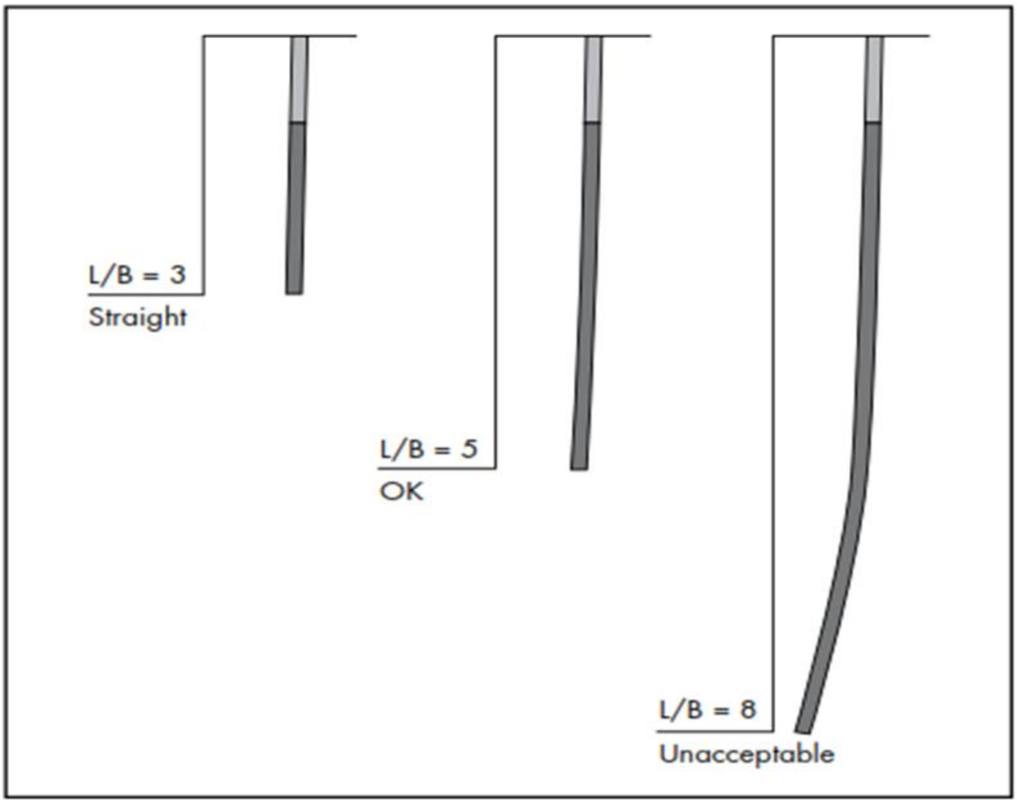

Figure 37 Effect of Drill Deviation on Increasing Face Height

Source: Darling Peter, 2011

As can be seen in the **figure 37**, drill deviation is unacceptable during blasting in open-pits when the bench height is excessive assuming a constant burden is used.

Another aspect that can be observed in *figure 40* is that when the bench height is excessive which results in drill deviation the blast-hole comes very close to the face as can be seen in the figure when L/B = 8, compared to lower values of L/B = 5 or L/B = 3 in which the blast-hole is away from the face, and when the blast-hole comes very close to the face its where most likely excessive fly-rock and air-blast problems occur with that blast as discussed earlier.

According to practice of blasting operations in open-pit mines, the recommended and good range for face height is 100 to 120 times the hole diameter and by going to extremes, for example, 200 times results in problems and it is also recommended that the excavation equipment should not dictate the face height unless local regulations control the face height (*Worsey P., 2011*).

III. Burden (B) and Spacing (S)

As regards to bench blasting there are guideline used within the surface mining industry dealing with burden and spacing as a multiple of hole diameter in bench blasting and **Table 4** shows such guidelines;

Table 4 Burden and Spacing as Multiple of Hole Diameter in Bench Blasting

Source: Darling Peter, 2011

	ANFO		Blends and Emulsions	
	Metric	U.S.	Metric	U.S.
Burden	× 25	× 24 (2 ft/in.)	× 30	× 30 (2.5 ft/in.)
Spacing	× 35	× 36 (3 ft/in.)	× 45	× 42 (3 ft/in.)

Although different equations may be used to arrive at the burden and spacing as a multiple of hole diameter relationship, the results are the same as shown in *Table 3*.

Basically there are two specific gravity divisions for the modern bulk of explosives; 0.8 to 0.87 for ANFO (*Ammonium Nitrate Fuel-Oil*) and 0.65 to 1.3 for emulsions and blends and as observed in table 1, two sets of burden and spacing rules are used that give a *powder factor* of approximately 0.6 kg/m^3, however, these are for rock which have **average strength** and where the required level of fragmentation is **medium to coarse**.

It is accepted common practice in bench blasting to use on a staggered pattern a burden-to-spacing ratio of around 1.15 to 1.4 for bench heights which are greater than four times the burden, the reason being, that the repeatable faces are developed with equal burden on both faces of relief, as will be illustrated in **Figure 38.**

Figure 38 shows the plan view of the geometry for a burden-to-spacing ratio of 1.4, where the Spacing (S) = 1.4 times the Burden (B), **S = 1.4B**, also showing the *Breakout Angle* and the burdens for each face being equal (*Lusk B., 2011*).

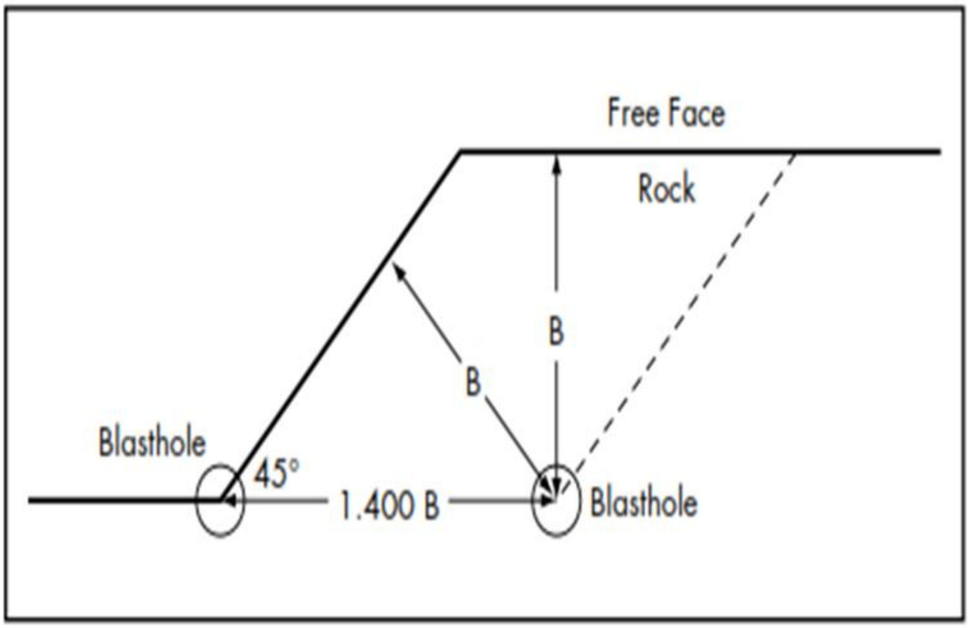

Figure 38 Geometry for a burden-to-spacing ratio of 1.4, S = 1.4B with Breakout Angle & burdens for each face being equal

Source: Darling Peter, 2011

IV. **Sub-Drilling**

Sub-drilling is required during the blasting of massive rock where there is no suitable horizontal bedding plane to maintain the floor grade. In rocks like granite sub-drilling is a must in order to maintain the grade without which the floor rises. The accepted amount of sub-drilling in surface mining practice is one-third (1/3) the burden, however, in some operations it may range from 0.2 to 0.5 times the burden (*Morhard et al. 1987*), but 1/3 the burden is a good guideline.

Sub-drilling can also be calculated as 5 to 8 times the hole-diameter, but the engineer should be aware that excessive sub-drilling causes smashing of the bench below and this makes drilling difficult and in general requires extra stemming to hold the later blast in place.

Looking it from another angle, that the excessive explosive energy from sub-drilling has nowhere to go, hence it becomes *wasted energy* and also creates extra ground vibrations. The negative effects of sub-drilling are controlled by offsetting the holes from one bench by half (1/2) the spacing and half (1/2) the burden when laying out the next bench. This hole-offset also reduces the risk of drilling into misfired "butt" (*Lusk B., 2011*).

In coal mining, when blasting the overburden it is common and appropriate to use *negative sub-drilling;* where the drill-hole stops above the coal to prevent coal loss and excessive fines.

V. **Stemming**

Stemming is used to confine the explosives column in order to prevent explosive energy from escaping and thereby reducing the effectiveness of the blasting.

In some cases the stemming is ejected prematurely this translates to the lost energy being costly but also to a generation of a sonic boom which also happens along with the release of fly-rocks.

But the downside is that stemming reduces the amount of explosives that can be placed in the hole, hence, reducing the *utilisation factor* of the hole.

An example to elaborate this aspect is when large diameter blast-holes have short benches then the hole utilisation can be 60% or even less but benches with a normal height-to-hole diameter ratios the hole utilisation is 85% or higher (*Morhard et al. 1987*).

In order to minimise the length of the blast-hole that contains the stemming, the stemming material is carefully chosen for its size and gradation. The optimum stemming material should have a diameter of around one-eighth (1/8) the hole diameter for small to medium diameter holes, this is so, to maximise the potential for interlocking and to also avoid *bridging* when pouring the explosive material into the hole, for instance, a 6 in. (150 mm) hole would require a 3/4 in. (19 mm) stemming and the stemming material locks good when it is "*clean"* meaning when it is free of fines rather than it is well graded (containing fines).

Hence, the guidelines for stemming can be expressed in an equation as follows;

a) For clean stemming with 1/8 of the hole diameter;

$$T = \frac{2}{3}B \tag{6.1}$$

b) For drill cuttings;

$$T = \frac{4}{3}B \tag{6.2}$$

T – Stemming Height

B – Burden

With these guidelines, it's important to keep stemming for large-diameter holes because the large face length at the top of the blast may have no explosives to fragment them, this produces serious over-size (*Lusk B., 2011*).

VI. Powder Column Length

This is also another vital parameter to take into consideration when designing a blast in open-pit mines and it is basically the amount of hole left after subtracting the stemming and the ***Hole Utilisation*** is the percentage of the hole used for powder. Normal hole utilisation values of 80% or higher should be expected (*Worsey P., 2011*).

VII. Decking

Decking is the separation of the explosives column in a blast-hole into two or more parts with stemming between them. It is common practice and recommended that the thickness of the deck material be 6 times the hole-diameter for dry holes and 12 times the hole-diameter for wet holes.

Decks are used in blasting for the following reasons (*Nemours E.I., 2005*):

1. To fill voids so that excessive explosives are not used. Even medium-sized voids can result in the excessive concentration of explosive energy. It is normal to measure the rise of decking in the hole until a normal rise is established and then to reprime and continue loading. In the case of voids, the same in-hole delay is used for each deck and a mud seam or other weak spot in the rock column has to be decked through to avoid fly-rock incidences.
2. To reduce the kilograms per delay when blasting close to residential structures to ensure that regulated limits are not exceeded. In this case, the stemming deck is used to divide the explosive column into two smaller decks. A delay of typically 25 ms is used between decks so that two small “thumps” are provided rather than one large one.
3. To reduce the amount of explosive in the hole. Air decks are used where an air void is placed either above or below the explosive. It is a way of decreasing the explosive load without the use of excessive stemming, which tends to lead to blocky ground on top of the pile and oversize.

Figure 39 shows how decking is applied in open-pit mine blasting;

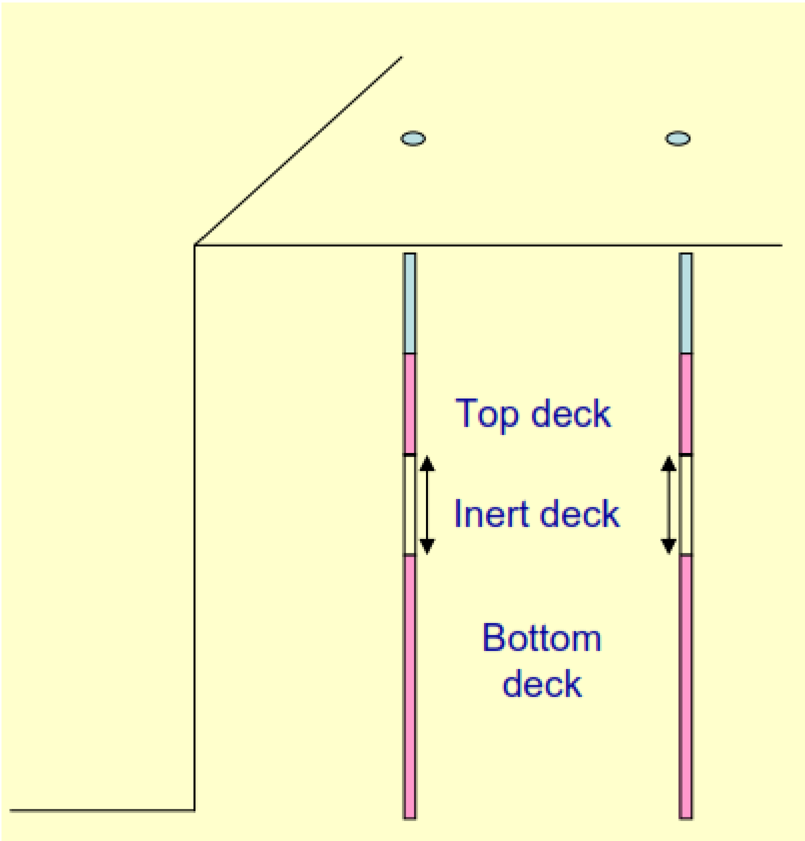

Figure 39 Decking in Open-Pit Rock Blasting

Source: Best D., 2008

VIII. **The Powder Factor**

The powder factor is usually determined by taking the amount of rock to be fragmented to grade and dividing it by the weight of explosives used and can be calculated for each hole or for a complete shot per round. In some mining operations, the powder factor is often expressed as the weight of explosive per unit weight of rock (kilograms per metric ton – kg/t); however, in some quarry markets in the United States, the powder factor is expressed in tons per pound (t/lb.), but in construction, the weight of explosive per unit volume of rock is used (kilograms per cubic meter – kg/m^3). There is a difference between *construction blasting* and *mining blasting*, construction blasting; the excavated volume is the quantity of significance, while the excavated ore tonnage, is of concern in the latter.

The *volumetric powder factor* is used in mining operations in connection with contract excavation overburden removal and waste stripping and for normal surface-mining operations, such as quarrying, a powder factor of 0.6 kg/m^3 is a good initial estimate.

The weight of the explosive (W) is calculated using a formula a follows;

$$W = PC \times C_\rho \tag{6.3}$$

PC – Powder Column Height

C_P – Column Density

The Column density (C_P) depends on the explosive diameter and explosive density and can be found using a manufacturer's chart or using equation 6.4:

$$C_\rho = \frac{(D_e^2 \times \rho)}{1.275\ kg/m} \tag{6.4}$$

D_e – Diameter of the Explosive

ρ – Density of Explosive

Equations 6.3 and 6.4 are both approximations due to the diameter of the drill hole being often larger than the diameter of the drill bit and the drill bits also wear considerably over their lifetime, therefore, it is usually impossible to measure the diameter of the hole at the depth of the explosive column (*Blaster's Handbook, 15th Edition, Wilmington Inc.*).

Hence;

$$Powder\ Factor = \frac{Weight\ of\ Explosive\ Used\ (kg)}{Weight\ of\ Rock\ (t)} \qquad \boldsymbol{or}$$

$$Powder\ Factor = \frac{Weight\ of\ Explosives\ Used\ (kg)}{Volume\ of\ Rock\ (m^3)} \tag{6.5}$$

IX. The Choice of Patterns

It is possible to choose a number of different patterns that include square, staggered, en echelon, and diamond, however, it is normal to use a staggered pattern in bench blasting, firstly, because the hole in the row behind is blasting into more solid rock rather than a weakened pocket and, secondly, because holes tend to break at 45° to the free face, making a square end to a bench almost impossible as shown in **figure 40**;

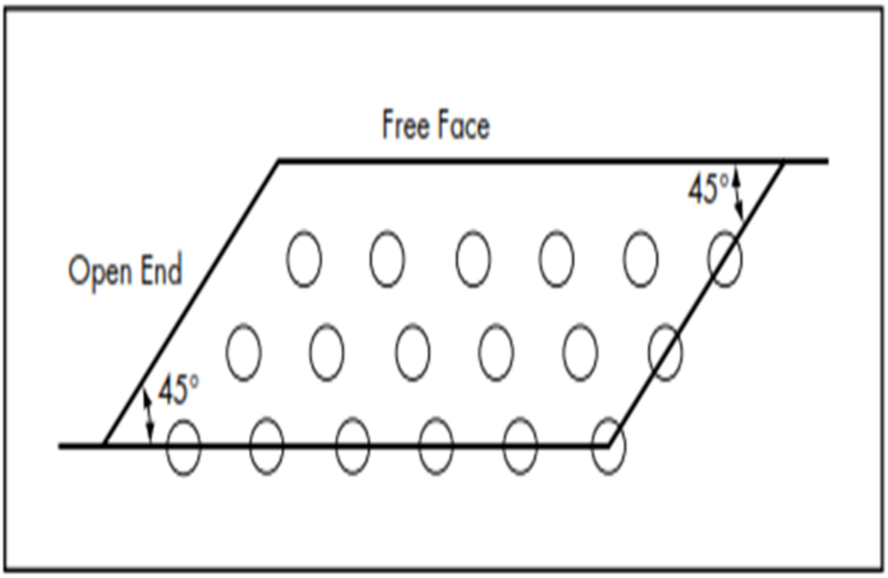

Figure 40 A Staggered Pattern with 3 rows having 6 holes per row

Source: Darling Peter, 2011

In some instances staggered patterns are avoided where angled holes are utilised. The staggered pattern may create logistical problems with drilling as the drill operator is forced to realign the boom angle rather than backing up perpendicular to the face and drilling subsequent rows using the same angle.

For a confined shot a square pattern is more recommended. A square pattern may also be recommended in sinking cuts when developing a new level. In construction blasting, a square or rectangular pattern helps maintain straight sidewalls, especially in road cuts (*Abdul H., 2013*).

X. The Number of Rows

The number of rows are also very important and depend on what the engineer want to achieve. For small production shots, a single row may be recommended depending on whether it is easy to maintain good face profiles. If it is difficult to maintain good face profiles, then multiple-row shots imply fewer drilling and loading issues. For blasting in quarries, it is normal to shoot three rows or more, with only special loading consideration necessary for the front row. Multiple-row shots also give workers less exposure to high-wall falls and failures. If employing more than three rows, the blast starts “piling up on itself,” resulting in a high muck-pile unless the timing pattern compensates for this or otherwise to compensate for this, a higher powder factor must be applied. Excessive rows without the compensation of an increased powder factor often result in excessive back break, especially at the crest of the bench, and a muck-pile that may eventually be higher than the original face height. Taking the number of rows to the extreme, one has a trench shot, which requires a powder factor of at least 1.2 kg/m^3 for successful fragmentation and reasonable digability (*Lusk Braden, 2011*).

Therefore in general, there are two main types of blasting patterns in open-pit mines which are;

a) Single Row Pattern
b) Multi-row Pattern

However, other blasting patterns also exits which utilise multi-sequence firing; examples of such patterns are;

i) Transverse Cut Pattern
ii) Wedge / Trapezoidal Pattern
iii) Diagonal Pattern

Single Row Pattern

A single row has low degree of fragmentation and has a high specific explosive consumption and for this reason and especially when the amount of blasted rock is huge, a single row pattern is not recommended for large mines.

Figure 41 shows a single row pattern;

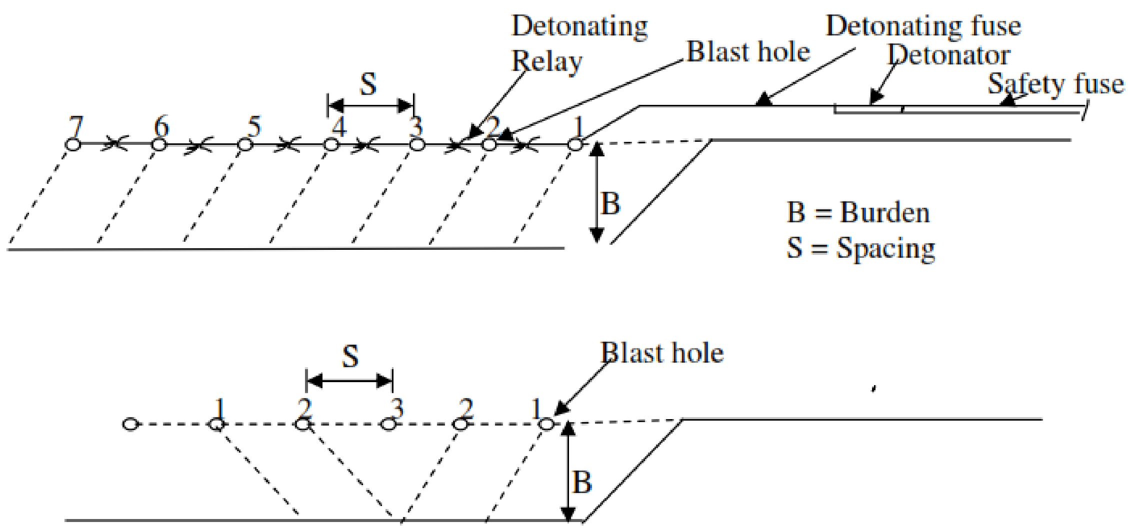

Figure 41 Single Row Blasting Pattern

Source: Darling Peter, 2011

Multi-Row Pattern

There are five types of multi-row patterns used in rock blasting in open-pit mines;

a) Square Grid In-line Initiation (spacing (S) = effective burden (B_E))
b) Square Grid V Pattern (effective spacing (S_E) $= 2B_E$)
c) Square Grid V_1 Pattern ($S_E = 5B_E$)
d) Staggered Grid V Pattern ($S_E = 1.25B_E$)
e) Staggered Grid V_1 Pattern ($S_E = 3.25B_E$)

Figures 42 and 43 show Multi-row blasting patterns;

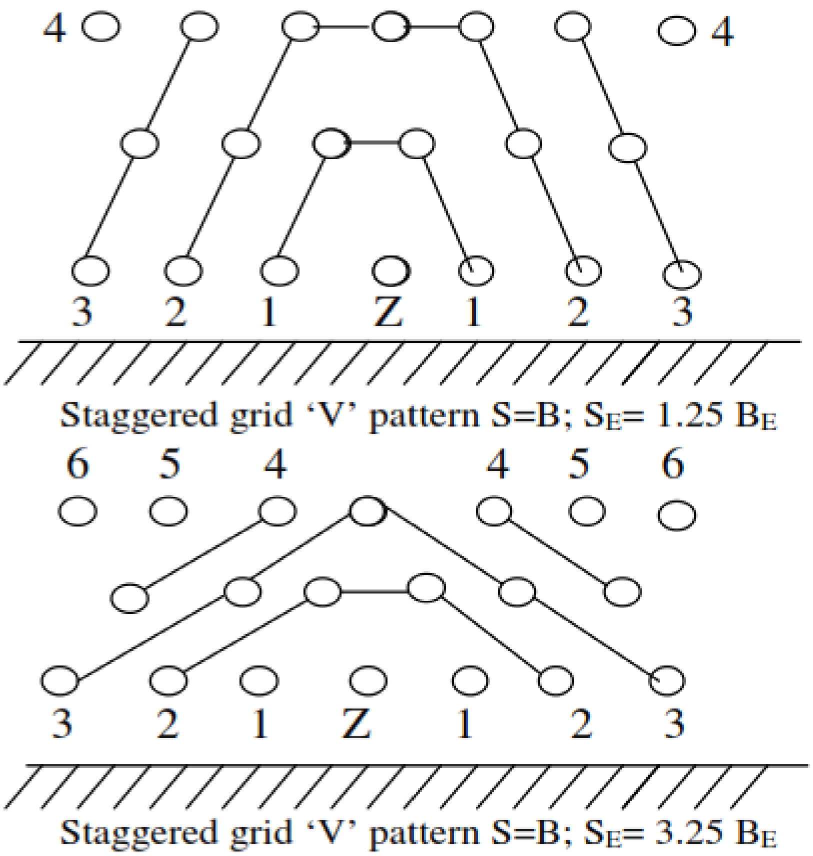

Figure 42 Multi-row Blasting Pattern Part 1

Source: Darling Peter, 2011

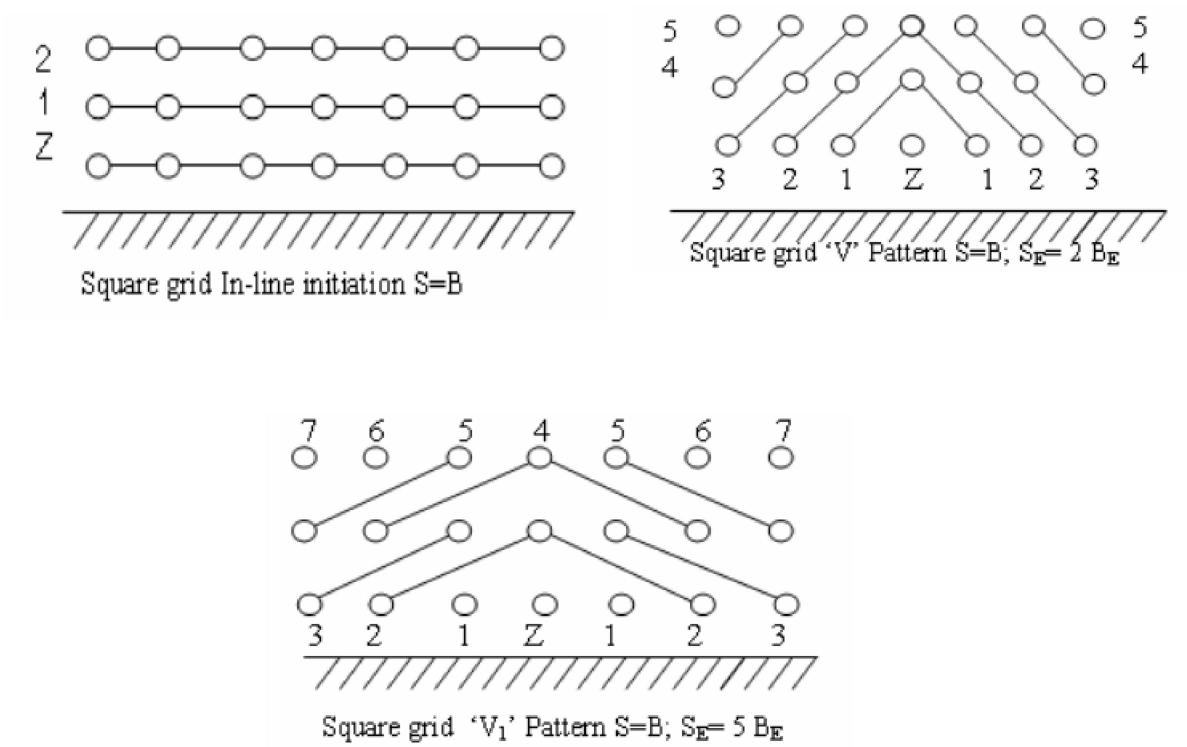

Figure 43 Multi-row Blasting Pattern Part 2

Source: Darling Peter, 2011

XI. Delays

There are two main types of delay in a blast pattern. These are;

- The hole-to-hole (inter-hole) delay
- The row-to-row (inter-row) delay

The optimum hole-to-hole delay for fragmentation has been determined by a number of different researchers, including the U.S. Bureau of Mines (*Stagg and Nutting 1987*). The general agreement is that the delay is **3 ms/m** of burden but maximum fragmentation is not desired all the time and in some instances the maximum throw is of primary importance, such as in blast casting e.g., the explosive casting of overburden from above a coal seam.

Figure 44 shows the effect of hole-to-hole delay on fragmentation where fragmentation is expressed as the ratio of the median fragment size (K50) for zero delay to that of increased delay between holes. The data were combined from five different sources (*Lusk B., 2011*);

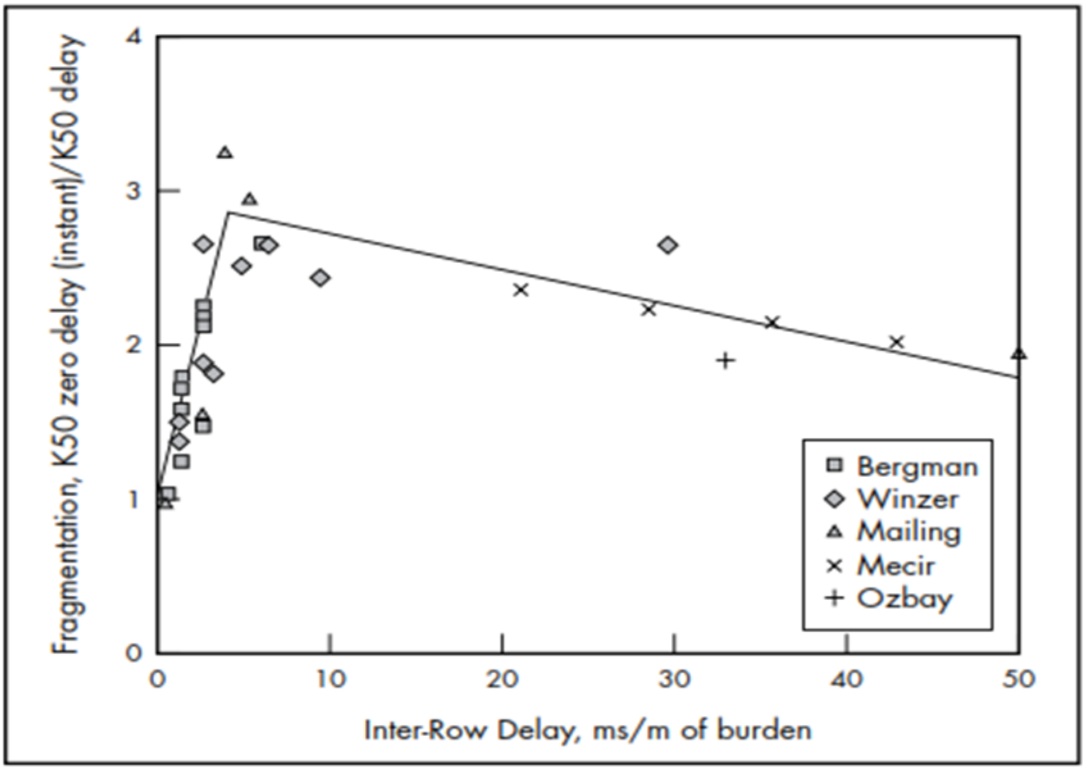

Figure 44 Effect of Hole-to-Hole Delay on Fragmentation during Blasting

Source: Grant 1990

As can be seen in **Figure 44**, fragmentation improves up to a delay of approximately 3 ms/m of burden and then gradually worsens and when using delays, the accuracy of the delay time is also very important.

Figure 45, shows how delay timing with conventional detonators is subject to scatter, and how accurate detonation delay times are most likely to be achieved when using electronic detonators. Due to the potential for inaccuracy in detonator timing, it is recommended to use a delay interval within a row larger than the 3 ms/m optimum. Large delay intervals between holes result in reduced fragmentation (*Worsey P., 2011*);

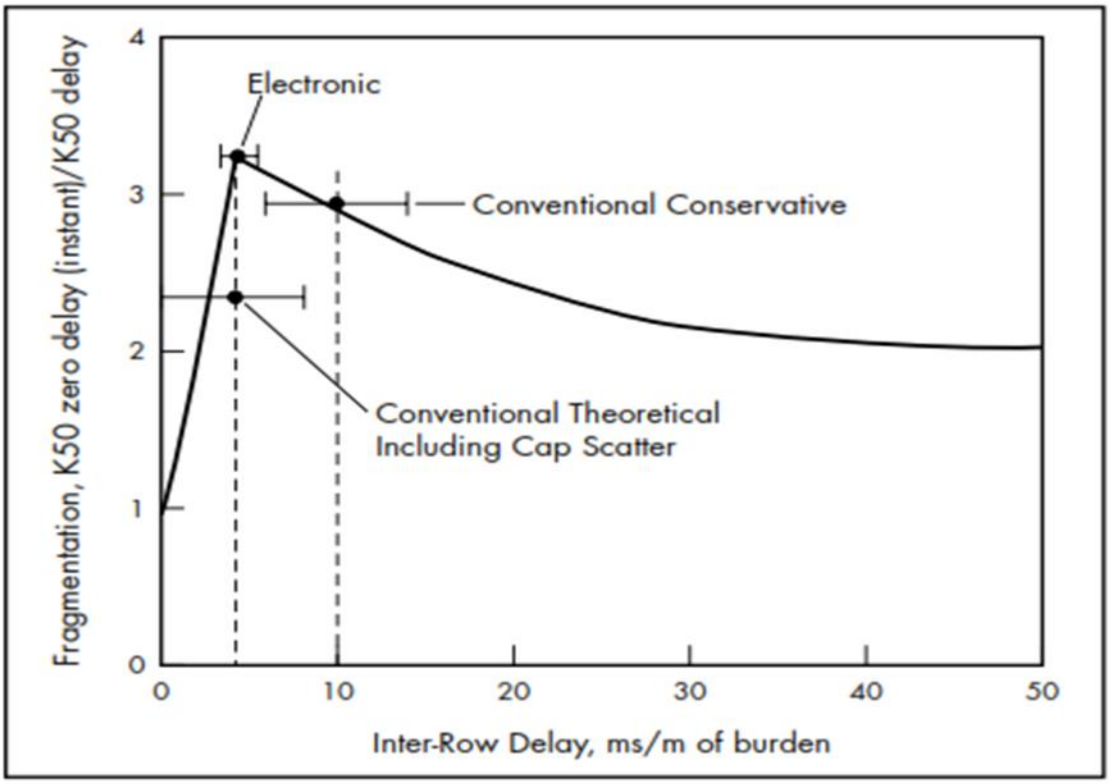

Figure 45 Effect of Inaccuracies in Timing on Fragmentation, showing the need for increased delays using conventional initiation systems

Source: Grant 1990

Inadequate delay between holes is also not recommended and basically results in *excessive throw*, increased back break/wall damage, and inadequate fragmentation (**seen in Figure 46**).

The firing of neighbouring holes together results in splitting of the rock between them and the premature propulsion of the rock mass forward which results in poor fragmentation.

The adverse loss in fragmentation before the "*sweet spot*" means surplus energy, which is consumed in throw, as shown in **Figure 45**. The row-to-row delay to provide good movement and fragmentation is a minimum of 3 ms/m of burden. As the number of rows increase to more than three, the value should be increased in order to provide good movement and to reduce the back break. Often significant "*back splatter*" (rock thrown backward on the bench) is a sign of insufficient delay between rows or excessive burden distance (*Worsey P., 2011*);

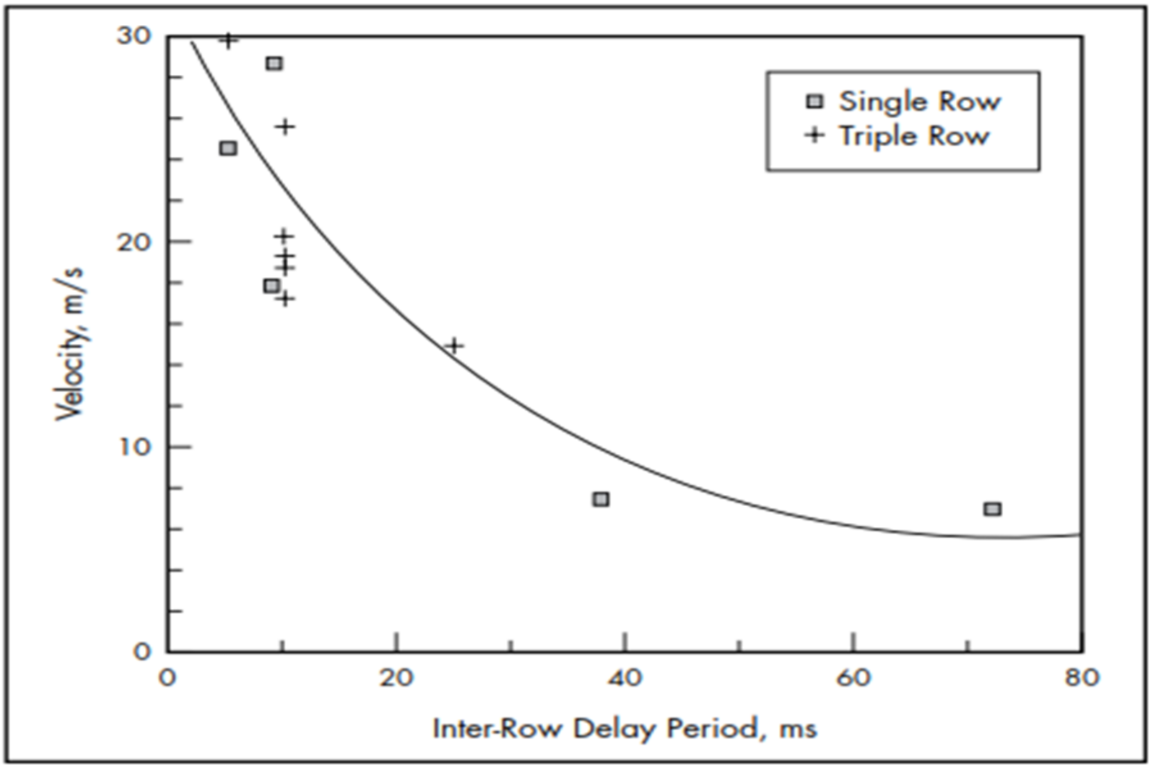

Figure 46 Effect of hole-to-hole delay on Face Velocity

Source: Grant 1990

Usually in rock blasting, it is normal to use a spacing-to-burden (S/B) ratio of 1.15:1 to 1.4:1, this implies that delays are a necessity for rock fragmentation but a penalty in blast casting because the burden and spacing are often reversed to achieve less fragmentation but more throw. This *splitting effect* is shown in **Figure 47**;

When holes are fired simultaneously with spacing-to-burden (S/B) ratios less than 2:1 there is a *splitting effect*, as can be seen in **figure 47**, where the fracture zones coalesce and provide a split before the actual fracturing reaches the free face (*Worsey Paul, 2011*).

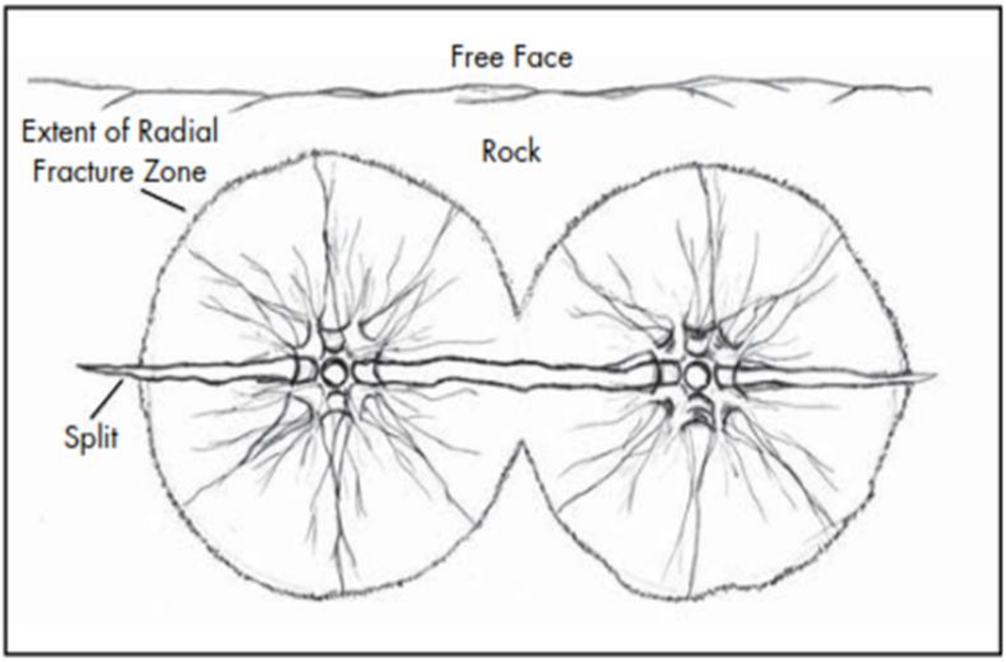

Figure 47 the Splitting Effect

Source: Darling Peter, 2011

7

BLASTING PROBLEMS AND SOLUTIONS

7.1 Introduction

In any blasting operation either in an underground mine or a surface mine there is bound to be blasting effects which may cause offsite harm and it is the duty of the engineer at site to ensure that offsite blasting effects are well controlled to prevent the hazard effect affecting people in the near vicinity and the environment.

This dissertation deals with the major problems associated with rock blasting in open-pit mines which can be seen video recordings of blasting operations itself or in captured images.

However, in general, the major problems associated with rock blasting in open-pits are;

a) Excessive Dust
b) The Presence of NO_X gases during blasting
c) Induced Ground Vibrations
d) Excessive Air blasts
e) Fly-rocks
f) Misfires
g) Over-fragmentation
h) No fragmentation (Detonation Failure)
i) Noise
j) Faulty Stemming and Escape of Explosive Energy

In this chapter noise and induced ground vibrations will not be examined further as the two cannot be directly observed either in a video recording of the blasting operation or captured image. Detonation failure and Over-Fragmentation have been recommended for future work.

7.2 Fly-rocks

When looking at the concept of fly-rocks in rock blasting it is important to define the term "*Blast Area*", and according to the United States code of Federal Regulations Title 30, *Blast Area* is the area in which the shock wave, flying material and gases from an explosion may cause injury to people.

During rock blasting the blast area is calculated using the following factors;

i. Type and amount of explosive
ii. Powder Factor, Delay system
iii. The type and amount of stemming
iv. Geology of rock
v. Blast pattern
vi. The Burden and angle, depth and diameter of blast-holes
vii. Experience in blasting of the mine operators

Hence, Fly-rocks are rocks which are propelled beyond the blast area during rock blasting.

7.2.1 Primary Causes of Fly-rocks

Fly-rocks present a dangerous situation to mine workers during rock blasting, therefore it is always important to secure the blast area and apply solutions that prevent and control fly-rock incidences during blasting.

Vast research has been done to try and identify the causes of fly-rocks during rock blasting and the majority of research in this area suggests that when there is a mismatch between the explosive energy and the mechanical strength of the rock being blasted fly-rocks are generated (*Mowrey G.L.*).

The following factors contribute and lead to the mismatch between rock strength and explosive energy;

a) Secondary blasting of boulders and toe holes
b) Wrong loading and firing practice
c) Deviation of blast-holes as discussed previously in chapter 6.4 on Drill Deviation
d) Insufficient delay between blast-holes in the same or different rows
e) A sudden decrease in rock resistance as a result of fracture planes in rock, geological faults, voids in rock, localised weakness in rock mass, mud seams in rock, etc.
f) High explosive concentration resulting in high energy density in the area where the explosive concentration is high

Figure 48 shows fly-rocks being propelled, the image has been produced by editing a video from a rock blasting operation at an open-pit mine in Canada (video number S1).

Video list and further details are in the Appendix.

Figure 48 Video S1 Paused at 00:00:57.16 showing Fly-rocks being propelled

(Further Details in Appendix)

Figure 49 shows further fly-rocks during a rock fragmentation operation at a surface mine;

Figure 49 Excessive Fly-rocks at a Mine Site

Source: Best D., 2008

7.2.2 Solution to Fly-rocks

- By frequently checking the rise of the powder column as this prevents overloading due to loss of powder in cracks and holes. Overloading causes the release of excessive energy which results in a mismatch between the strength of the rock being blasted and the increased explosive energy.
- In high-wall faces engineers should either physically examine the high-wall face for zones of weaknesses, faults, back-breaks, concavity (in-ward curving) or use laser scanning to examine for these parameters as they tend to reduce the burden during blasting resulting in fly-rocks due to insufficient burden.
- By using a Bench Height to Burden Ratio $\left(\frac{L}{B}\right)$ between 3 to 5, this is the recommended face aspect ratio as it prevents blast-holes deviating close to the face
- The presence of rocks on the top of the bench also causes fly-rocks during blasting as these rocks are propelled beyond the blast area, hence it is necessary to ensure that the top of all benches are clean and free of rocks.

7.2.3 Calculating the Fly-rock Zone

Calculating the fly-rock zone is important as well, because it allows the engineer to demarcate the risk zone, define the blast area and calculate the risk involved in which probabilistic models may be used. Empirical formulae exist that can be used to calculate the fly-rock zone, however, there are some restrictions with this approach; at present empirical formulae that exist can be used to calculate the throw distance and the initial velocity of rock fragments but these formulae are only for granite and when applied for other rocks they are not accurate as the other rock is thrown at a greater distance than the predicted distance during blasting.

Today, with advancement in computer technology, Artificial Intelligence (AI) can be used to better predict the fly-rock zone, this approach at present is the most accurate way of calculating the fly-rock zone.

What Artificial Intelligence does is incorporate a given model and then produce a simulation which is carried multiple times to give the best prediction possible in accordance with the values that are fed into the computer and the mathematical model itself.

There are three fly-rock mechanisms which are considered in developing a model for predicting the throw distance, these include;

- **Face Burst** – This is due to burden conditions which influence the fly-rock distances in front of the face
- **Cratering** – It happens when the stemming height to hole diameter ratio is smaller than recommended or the collar rock is weak hence the fly-rocks can be projected in any direction from the crater at the collar of the blast-hole.
- **Rifling** – when the height of stemming is within recommended limits and still fly-rocks are being projected, this may be due to *Rifling;* which is the release of stemming material and loose rocks from the collar, selecting inappropriate stemming material may cause this effect (*Richards A.B., Moore A.J., 2005*).

The **figure 50** shows the three fly-rock mechanisms;

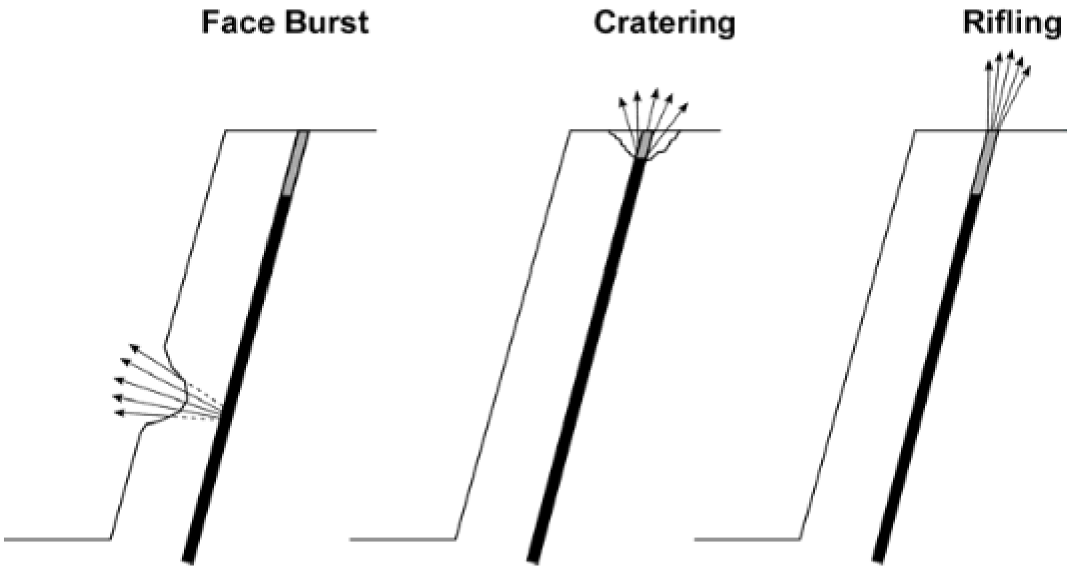

Figure 50 Main Fly-rock Mechanisms

Source: Moore A.J., 2005

In 2005 a company in Australia, **Terrock Consulting Engineers Pty Ltd** developed a Predictive Model to calculate the Maximum Throw of fly-rocks during blasting and to be applied in open-pit mines. Data that was used in developing the model was collected from Kalgoorlie Mines from 29th October to 2nd December in 2004.

The predictive model is used by engineers during planning and assessment when demarcating the fly-rock zone in order to establish the blast area which is very vital for safety reasons during rock blasting.

The model uses the following as inputs; site calibration factor, charge mass per metre, the burden and the stemming height.

The Model for Fly-rock Throw is;

$$L = \frac{k^2}{9.8}\left(\frac{\sqrt{m}}{SH}\right)^{2.6} \sin 2\theta \qquad (7.0)$$

L – Fly-rock Throw (m), L_{max} – Maximum Fly-rock Throw

m – Charge Mass per Delay (kg)

SH – Stemming Height (m)

θ – Horizontal Launch Angle

k – K-Constant and when ANFO is used k = 21.9 for the sulphide zone blasts and k = 28.3 for oxide zone blasts. When ENERGAN explosive is used $k < 17.2$ for sulphide zone blasts and $k < 22.3$ for oxide zone blasts.

Because the Maximum Throw occurs when the $\theta = 45^0$, hence;

$$L_{max} = \frac{k^2}{9.8}\left(\frac{\sqrt{m}}{SH}\right)^{2.6} \qquad (7.1)$$

Using this model for two explosives ANFO and ENERGAN, **Terrock Consulting Engineers Pty Ltd** were able to come up with charts that show the relationship between the fly-rock distance and the stemming height both in metres. The charts prove a very important factor that by controlling the stemming height, the projected distance of fly-rocks can also be controlled hence reduced in order to secure a safe blast area.

Figures 51 and **52** show these charts from this Predictive Model;

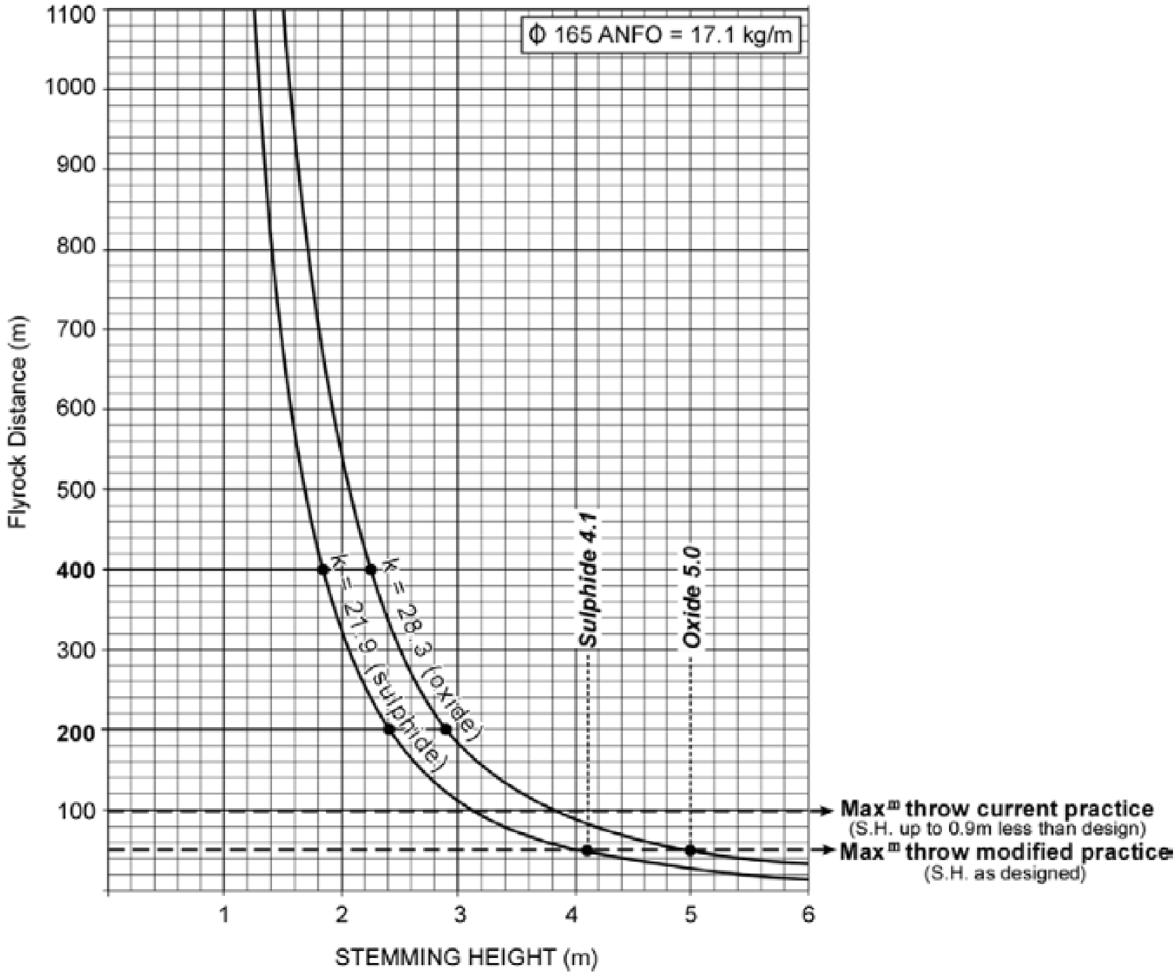

Figure 51 Influence of Stemming Height on Fly-rock Distance for ANFO

Source: Terrock Consulting Engineers Pty Ltd (Moore A.J., 2005)

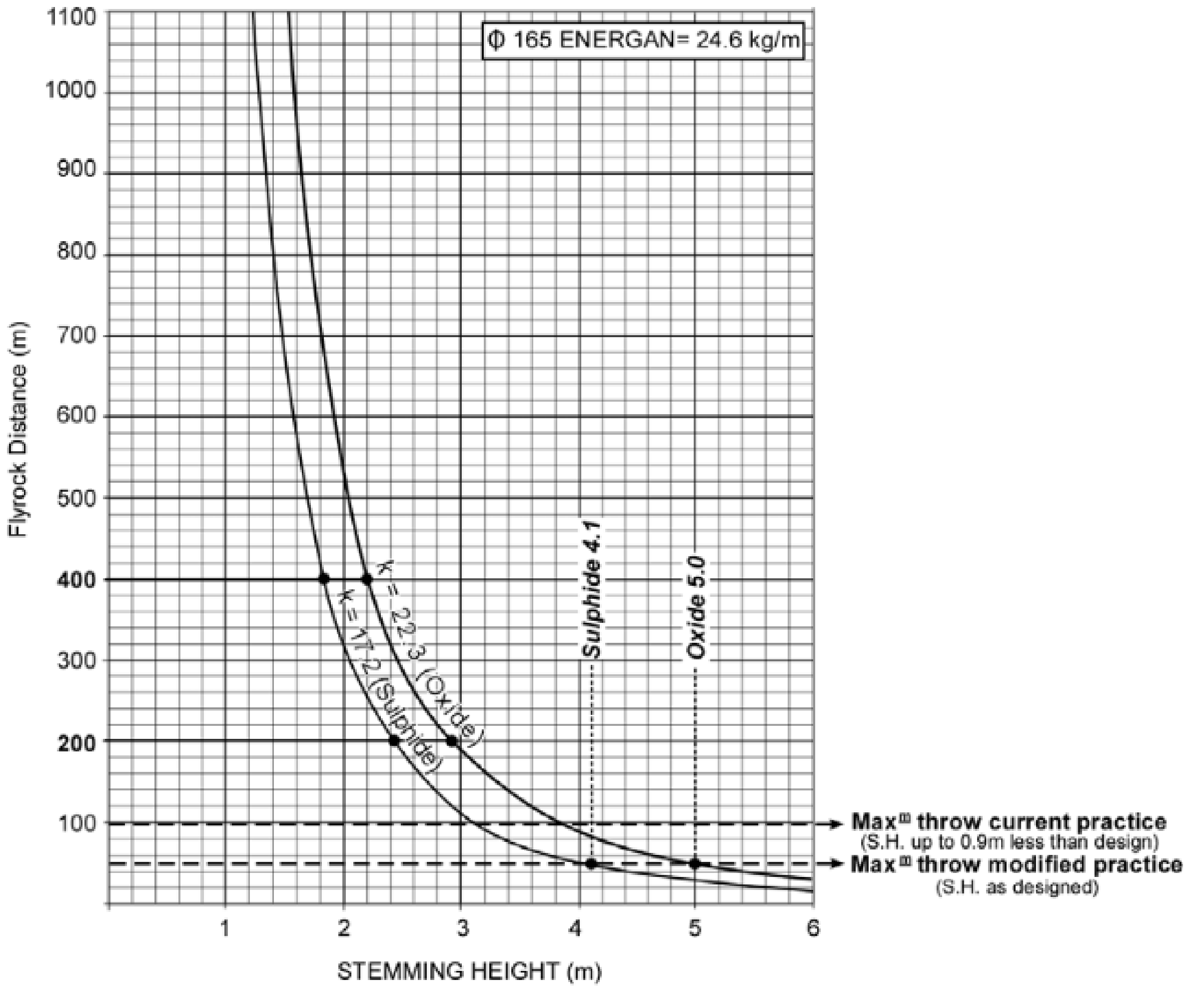

Figure 52 Influence of Stemming Height on Fly-rock Distance for ENERGAN explosive

Source: Terrock Consulting Engineers Pty Ltd (Moore A.J., 2005)

7.2.4 Calculating the Evacuation Distance

Work Safe Victoria, a Government agency in Victoria, Australia after doing research on safe distances in relation to the minimum stemming length recommended that the following values as shown in Table 4 be used as evacuation distances depending on the minimum stemming length used in open pit rock blasting. The data should be used if ANFO or Emulsion bends with a **charge density of 1.2 g/cc** common in small to intermediate blast-hole diameters are used; (**Table from Work Safe Victoria, 2009**);

Table 5 Recommended Evacuation Distances depending on the Minimum Stemming Length Used with Charge Density = 1.2 g/cc

- Small to intermediate blast hole diameter using ANFO/emulsion blends (1.2g/cc)									
Evacuation distance (m)	Minimum stemming length (m)								
	51mm	76mm	89mm	102mm	114mm	127mm	140mm	152mm	165mm
100	1.4	2.3	2.9	3.7	4.3	4.9	5.6	6.3	7.0
150	1.1	1.9	2.3	2.9	3.5	4.0	4.5	5.1	5.7
200	0.9	1.6	2.0	2.5	3.0	3.4	3.9	4.4	4.9
250	0.8	1.4	1.8	2.2	2.6	3.0	3.4	3.9	4.3
300	0.8	1.3	1.6	2.0	2.4	2.7	3.1	3.5	3.9
350	0.7	1.2	1.5	1.6	2.2	2.5	2.8	3.2	3.6
400	0.7	1.1	1.4	1.7	2.0	2.3	2.6	3.0	3.3
450	0.6	1.1	1.3	1.6	1.9	2.2	2.5	2.8	3.1
500	0.6	1.0	1.2	1.5	1.7	2.0	2.3	2.6	2.9
550	0.6	1.0	1.2	1.4	1.7	1.9	2.2	2.5	2.8
600	0.6	0.9	1.1	1.4	1.6	1.8	2.1	2.3	2.6
650	0.5	0.9	1.1	1.3	1.5	1.7	2.0	2.2	2.5
700	0.5	0.8	1.0	1.2	1.4	1.7	1.9	2.1	2.4
750	0.5	0.8	1.0	1.2	1.4	1.6	1.8	2.1	2.3
800	0.4	0.7	0.9	1.1	1.3	1.4	1.7	2.0	2.2
Note – shaded areas indicate increase risk of fly rock (stemming lengths less than 25 times the diameter of the hole)									

Data in **Table 5** can be used to define exclusion zones which help reduce the risk of injury due to fly rocks. The table uses stemming length and blast-hole diameter as inputs, with that the evacuation distance is obtained.

Work Safe Victoria calculated the values with a ***Safety Factor*** of 150% and this means that if the stemming length in all the blast-holes is equal to the recommended value in Table 4 then no fly rock should be thrown more than 2/3 the recommended evacuation distance.

As an example, when a charge density of 1.2 g/cc is loaded into 102 mm blast-holes and all the blast-holes have a stemming length of 2m, then the recommended evacuation distance should be 300 m but the actual fly rock theoretically will not be thrown more than 200 m.

Engineers should then plan for the exclusion zone after calculating the recommended evacuation distance. Planning measures should include;

a) Considering the site location and map reference
b) Outlining the exclusion zone and distance from the blast area
c) Outlining all the roads going in and coming out of the exclusion zone
d) Considering the location in terms of all facilities present i.e. houses, pipes, surface and underground utilities
e) Considering all facilities that are vulnerable including protected public works

Fine drill cuttings and rounded rocks sometimes are not good materials for stemming, through common practice crushed angular rock is highly recommended as a suitable stemming material because crushed angular rock;

- Wedges against the sides of the blast-hole thereby resisting premature ejection
- Maintains frictional forces in wet blast-holes by displacing the water

7.3 Faulty Stemming and the Escape of Explosive Energy

When explosive energy escapes during detonation as a result of faulty stemming, the degree of rock fragmentation is affected by being reduced hence the desired fragmentation level is not achieved. **Figure 53** illustrates the escape of explosive energy due to faulty stemming;

Figure 53 Video S13 Paused at 00:01:18.2 showing Energy Escape during Detonation

(Further Details in Appendix)

Figure 54 further shows explosive energy escaping during blasting at a surface mine;

Figure 54 Explosive Energy Escape during Detonation

(Photograph by Naoya Hatakeyama)

7.3.1 Solution to Explosive Energy Escape due to Faulty Stemming

The following practices should be applied to avoid faulty stemming hence control and prevent the escape of explosive energy;

- The optimum stemming material should have a diameter of around one-eighth (1/8) the hole diameter for small to medium diameter holes, this is so, to maximise the potential for interlocking and to also avoid bridging when pouring the explosive material into the hole, for instance, a 6 in. (150 mm) hole would require a 3/4 in. (19 mm) stemming and the stemming material locks good when it is "clean" meaning when it is free of fines rather than it is well graded (containing fines) as discussed in chapter 6.
- When using clean stemming with 1/8 of the hole diameter, it is recommended that the stemming height (T) should be two-thirds (2/3) the burden (B); hence T = 2/3B as in equation 6.1 in chapter 6
- With drill-cuttings, engineers are recommended to use the stemming height (T) equal to four-thirds (4/3) the burden (B) hence T = 4/3B as in equation 6.2 in chapter 6.
- Equations 6.1 and 6.2 also prevent over-size in large diameter blast-holes as discussed in chapter 6.
- Coarse, large and sharp rocks are not recommended to be used as stemming material as they damage the initiation system.
- When crushed rock is used as stemming material, the following dimensions represent common practice;

Blast-hole Diameter (in.)	Crushed rock size (in.)
1.5	0.375
2 – 3.5	0.375 – 0.5
4 – 5	0.625
≥ 5	0.75

7.4 Misfires and Prevention Techniques

The main causes of misfires have been discussed in Chapter 5, **figure 55** shows a site of a misfire photographed by the Mine Safety and Health Administration (MSHA) in the United States in June 2000. In the photograph, a misfire was accidentally struck by an air drill also shown in the figure, during a mining operation.

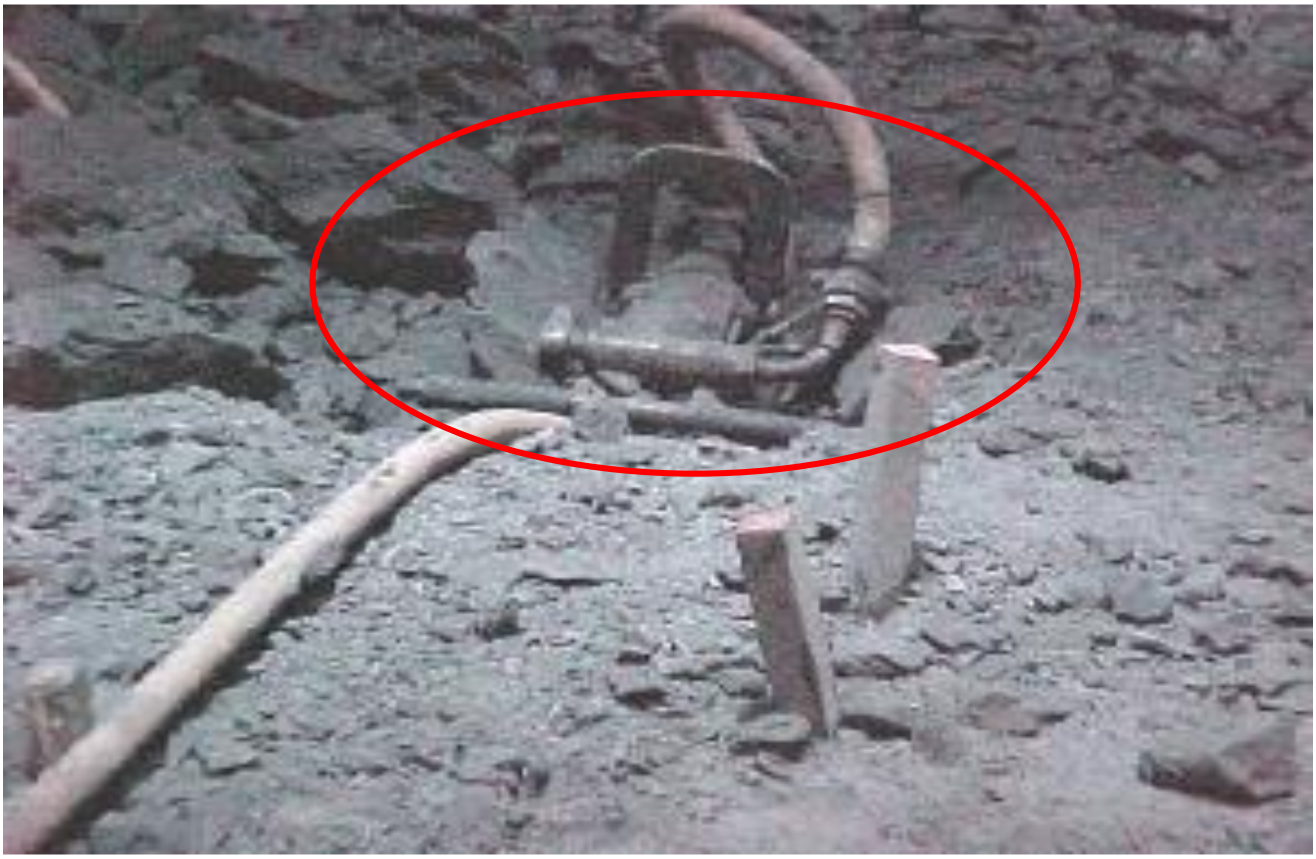

Figure 55 An Air Drill that Struck a Misfire

(Photograph by Mine Safety and Health Administration (MSHA) in U.S.)

The circled area shows an air drill after the accidental explosion.

The **figure 55** was part of an investigation, after an accident occurred involving a misfire in June 2000 in the United States District 8 of the Mine Safety and Health Administration (MSHA).

The details of the accident were as follows;

"*A serious explosives accident occurred recently at a District 8 area mine. The accident involved a new shaft and slope sinking project under development by a mining construction company. The accident occurred when one of the construction workers, utilizing an air drill, struck an apparent misfire from a previous day's shot. The non-detonated portion of the explosive charge consisted of one eight inch stick of blasting powder. Five construction workers were in the shaft when the blast occurred. Three persons sustained injuries from the force of the blast. Two men were treated at a local hospital, kept overnight, and released the next day. The third man received multiple injuries and remains in the hospital after undergoing surgery*."

Following the accident the MSHA made the following recommendations;

- *"That the company should ensure that adequate examinations for hazardous conditions in all workplaces prior to beginning any work*
- *That a qualified person should perform adequate and thorough examinations for misfires immediately after the blasting area has cleared*
- *When misfires occur, only work by a qualified person to dispose of the misfire, or to protect persons in the affected area should be performed."*

\- June 2000

\- United States Department of Labor; Mine Safety and Health Administration – MSHA

7.4.1 Management and Control of Misfires

The following practices and techniques should be applied in order to prevent and manage misfires at a mine site;

- Mine operators should carefully examine a site after every firing for misfires, specifically looking for noxious fumes, evidence of undetonated explosives, poor rock fragmentation and sometimes abnormal blast sounds.
- It is recommended that only qualified personnel should examine the site because it is possible for misfires to remain undetected after initial inspection hence regular inspection of the face and the muck pile is highly recommended.
- When blasting in critical geological environments simultaneous firing of all charges may be applied to prevent the rock shifting in unfired zone which damages the blasting fuse.
- Connections should be checked immediately before a blast to ensure the integrity of the initiation system and to minimise the risk of a misfire. Where in-hole initiation is used, i.e. with the detonator placed inside the hole, two detonators are needed for each deck or column of explosives to minimise the possibility of a misfire. This is because faulty detonators cannot be easily identified or recovered.
- It is highly recommended that shock tube connectors be covered with enough material to prevent damage to surface lines by shrapnel; about 200 mm of damp dust or chippings is usually enough.
- If the firing was effected electrically the circuit shall be tested again and an attempt made to refire the charges before it is approached by any person and if the attempt fails, the leads shall be disconnected from the exploder and ten minutes shall be allowed to elapse before the charge is approached and if the firing was effected by safety fuse, the charge shall not be approached by any person until not less than thirty minutes has elapsed since the lighting of the fuse.

- In case of a misfire, no personnel should enter the identified danger area, before 30 minutes have elapsed if the explosive shot was fired using a safety fuse and a blasting cap or before 5 minutes (***the MSHA recommends 15 minutes***) if other apparatus was used to fire the shot and after the shot-firing device has been removed. All cases of misfires should be recorded.
- It is very important that all misfires at the mine be investigated as this enables the cause of the misfire to be identified and helps in enacting measures that prevent future misfires from occurring.
- After a misfire has been identified, shot firers should immediately notify their supervisor, then wait for a specified period as stated above depending on which device was used to fire the explosive shot and the following steps should then be taken;
    i. Under supervision shot firers should disconnect the blasting cable from the power source
    ii. Shot firers are advised to short circuit the battery end before a closer examination of the electric connectors
    iii. Under supervision, shot firers should then remove permissible explosives by using wooden tools either by washing the stemming together with the permissible explosives from the blast-hole with water or by washing the stemming and then insert a new primer, after inserting this new primer, the explosive shot should then be fired again
    iv. However, there may be cases where the solution stated in (iii) might not practically work in removing the misfire, if this is the case, then under supervision, shot firers should remove the misfire by firing a separate charge at least 2 ft. (61 cm) away from the misfired charge and in parallel to the misfired charge, this will remove the misfire if solution (iii) does not work.

7.4.2 Misfire Risk Analysis Model

In 2004 the Norwegian University of Science and Technology (NTNU) and the Norwegian Tunnelling Society (NFF) developed a model for quantifying the *Risk* associated with accidental detonation of misfires.

The model is built using Probability Statistics and Set Theory Mathematics.

The goal was to be able to estimate the *Probability* of Accidental Misfire Detonation and the Risk of Fatal Injuries from these accidental detonations.

In Probability Statistics a Risk is;

$$\boldsymbol{Risk} = \boldsymbol{Likelihood} \times \boldsymbol{Consequence}$$

Where the Likelihood is either the Frequency or Probability.

The **Frequency** is taken as the rate at which events occur over time while **Probability** is the rate of a possible event which is expressed as a fraction of the total events and hence **Consequence** is either the direct effect of an event or incident (*Myers R.H., 1998*).

In Set Theory Mathematics, An Event is a subset of a sample space and the union of two elements, let's say A and B is expressed as $A \cup B$ and it is an event that contains all elements belonging to A or B or both and if A and B are two events, the Probability of A union B is;

$$\boldsymbol{P}\ (\boldsymbol{A} \cup \boldsymbol{B}) = \boldsymbol{P}(\boldsymbol{A}) + \boldsymbol{P}(\boldsymbol{B}) - \boldsymbol{P}(\boldsymbol{A} \cap \boldsymbol{B})$$

$\boldsymbol{A} \cap \boldsymbol{B}$ is the intersection of A and B and it is defined as all elements that are common between A and B and when there are no elements common between A and B, then $\boldsymbol{A} \cap \boldsymbol{B} = \boldsymbol{0}$ and the two events are said to be **Disjoint**.

When events A and B are disjoint then their union becomes;

$$\boldsymbol{P}\ (\boldsymbol{A} \cup \boldsymbol{B}) = \boldsymbol{P}(\boldsymbol{A}) + \boldsymbol{P}(\boldsymbol{B})$$

Another important aspect is *Conditional Probability*; when an event occurs and another event is known to have occurred, for example, the Conditional Probability of event B given that it is known that event A occurred is expressed as;

$$\boldsymbol{P}(\mathbf{B} \mid \mathbf{A}) = \frac{\boldsymbol{P}(\boldsymbol{A} \cap \boldsymbol{B})}{\boldsymbol{P}(\boldsymbol{A})} \quad \boldsymbol{if}\ \boldsymbol{P}(\boldsymbol{A}) > \mathbf{0}$$

Two events are said to be *Independent* when event A does not affect event B and vice versa, and the probability of the intersection of two independent events is;

$$\boldsymbol{P}(\boldsymbol{A} \cap \boldsymbol{B}) = \boldsymbol{P}(\boldsymbol{A}) \times \boldsymbol{P}(\boldsymbol{B})$$

The model also uses the **AND** & **OR** operators; all events in these two operations are independent;

<u>AND OPERATOR</u>

$$\boldsymbol{P}(\boldsymbol{E}) = \boldsymbol{P}(\boldsymbol{e_1} \cap \boldsymbol{e_2} \cap \boldsymbol{e_3}) = \boldsymbol{P}(\boldsymbol{e_1}) \times \boldsymbol{P}(\boldsymbol{e_2}) \times \boldsymbol{P}(\boldsymbol{e_3})$$

<u>OR OPERATION</u>

$$\boldsymbol{P}(\boldsymbol{E}) = \boldsymbol{P}(\boldsymbol{e_1} \cup \boldsymbol{e_2} \cup \boldsymbol{e_3}) =$$

$$\boldsymbol{P}(\boldsymbol{e_1}) + \boldsymbol{P}(\boldsymbol{e_2}) + \boldsymbol{P}(\boldsymbol{e_3}) - \boldsymbol{P}(\boldsymbol{e_1}) \times \boldsymbol{P}(\boldsymbol{e_2}) - \boldsymbol{P}(\boldsymbol{e_1}) \times \boldsymbol{P}(\boldsymbol{e_3}) - \boldsymbol{P}(\boldsymbol{e_2}) \times \boldsymbol{P}(\boldsymbol{e_3}) + \mathbf{2}\big(\boldsymbol{P}(\boldsymbol{e_1}) \times \boldsymbol{P}(\boldsymbol{e_2}) \times \boldsymbol{P}(\boldsymbol{e_3})\big)$$

Figure 56 shows the simulation code symbols of the Misfire Risk Analysis Model based on OR & AND Operators;

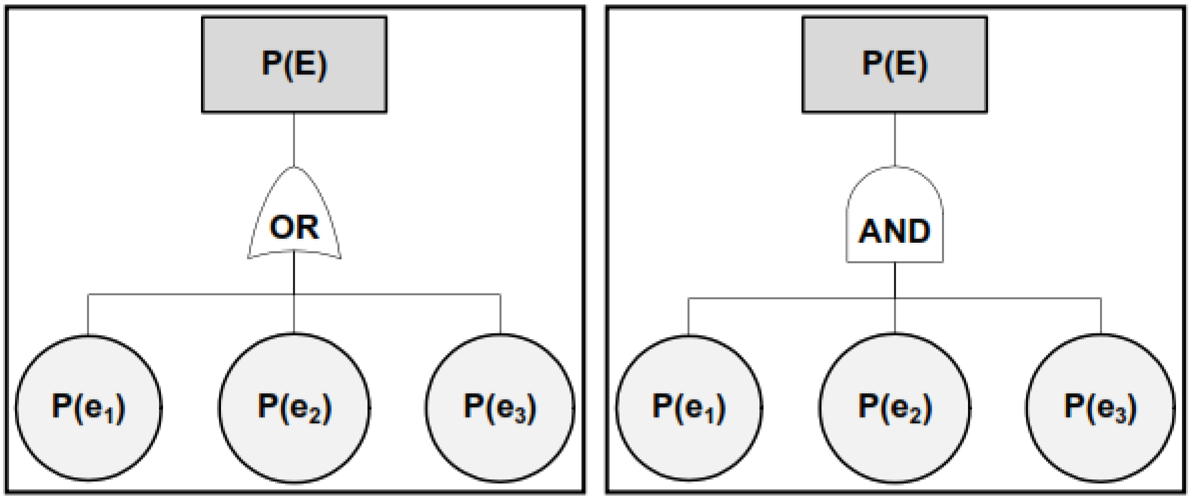

Figure 56 Simulation Code Symbols

Source: Olsen V., 2008

The Risk Analysis Model is built on the following assumptions;

a) A misfire must be present
b) The misfire must be hit
c) The misfire must detonate when hit

The research done by NTNU also found that a misfire will occur due to the following factors (*Olsen V., 2008*);

a) Defects in the explosive product
b) Error committed by shot-firers (workmanship fault)
c) Cut-off column charge
d) Dead pressuring

Figure 57 shows the developed model done at the Norwegian University of Science and Technology (NTNU); incorporates Fault Tree Analysis;

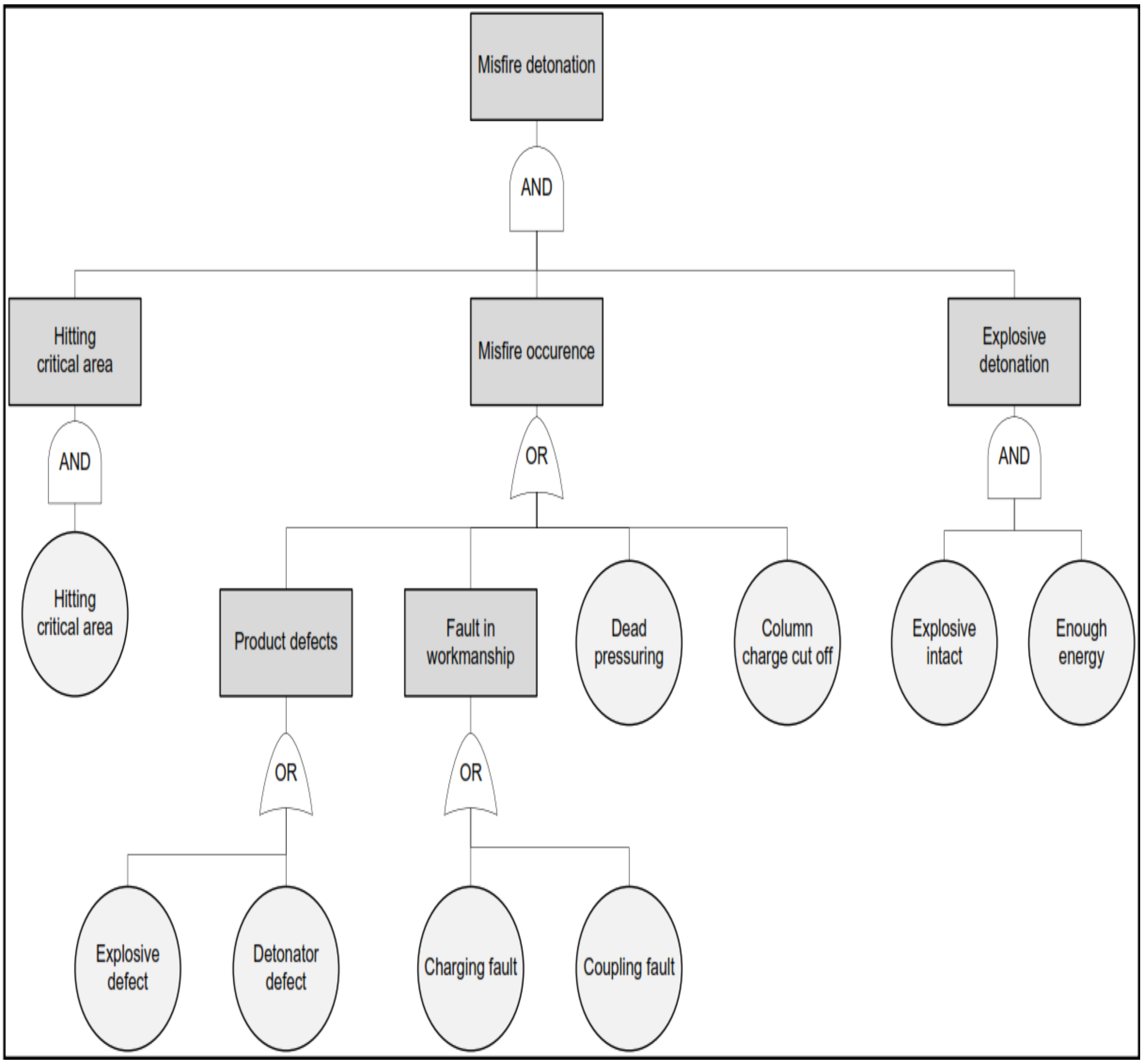

Figure 57 Misfire Risk Analysis Model at NTNU

Source: Olsen V., 2008

The following conclusions were made from the Misfire Risk Analysis Model;

1) **Probability that a Misfire will Occur P(MO)**

$$P(MO) = \frac{n_m}{n_{dh}}$$

$P(MO)$ – Probability of Misfire Occurrence

n_m – Number of misfires

n_{dh} – Number of drilled holes

2) **Probability that a Misfire will Detonate**

$$P(MD) = \frac{n_{md}}{n_{mh}}\,(1 - f)$$

$P(MD)$ – Probability of Misfire Detonation

n_{mh} – Number of misfire hits

n_{md} – Number of misfire detonations

f – Detection degree factor (drilling through rock, *f* = 0, cleaning the bench, *f* = 0.9)

3) **Probability of a Fatal Injury due to Misfire Detonation P(FI)**

$$P(FI) = \frac{n_{fa}}{n_{md}}$$

$P(FI)$ – Probability of fatal injury from misfire detonation

n_{fa} – Number of fatal injuries

n_{md} – Number of misfire detonations

4) **Probability of Hitting a Critical Area during Drilling $\boldsymbol{P(CA_d)}$**

$$P(CA_d) = \frac{A_m + A_b}{A_d} = \frac{A_{cm} Sin\, \alpha + A_{sm} Cos\, \alpha + A_b}{A_d}$$

$P(CA_d) -$ Probability of striking a critical area during drilling

A_m – Horizontal misfire exposure area

$A_b -$ Drill bit cross-section area

$A_d -$ The previous overlaying drilling pattern

$\alpha -$ Misfire angle during rock debris drilling ($\alpha = 90° =$ vertical direction)

$A_{sm} -$ Long side area of the misfire

$A_{cm} -$ Cross-section area of the misfire

Hence, the **Total Risk of Having a Fatal Injury from Misfire Detonation** is;

$$\boldsymbol{P(FA_{d1}) = P(MO) \times P(MD) \times P(FI) \times P(CA_d)}$$

Simplifying mathematically;

$$\boldsymbol{P(FA_{d1}) = \left(\frac{n_m}{n_{dh}}\right) \times \left(\frac{n_{md}}{n_{mh}}\right) \times \left(\frac{n_{fa}}{n_{md}}\right) \times \left(\frac{A_{cm} Sin\, \alpha + A_{sm} Cos\, \alpha + A_b}{A_d}\right) \times (1 - f)}$$

$P(FA_{d1}) -$ Total Risk of Having a Fatal Injury from a Misfire Detonation.

7.5 Excessive Air-blasts Problems

An Air-blast occurs when the explosive energy reaches the free face and is transferred into the air as a P-wave. An air-blast can be recorded in decibels (**dB**) and is often expressed in pounds per square inch (**psi**). **Figure 58** shows an air-blast photographed during blasting at an open pit mine site;

Figure 58 an Air Blast during Rock Blasting

(Photograph by Naoya Hatakeyama)

7.5.1 Causes of Excessive Air-blasts during Rock Blasting

The pressure wave of an air-blast creates a push (which is positive pressure) and a pull (which is negative pressure) effect. Sources of air-blast can be categorised as follows;

i) Noise Energy – High frequency waves from delays
ii) Stemming Release Pulse – When the stemming blows out creating a high frequency wave above an air pressure pulse
iii) Rock Pressure Pulse – Seismic activity at the base of the rock during fragmentation
iv) Air Pressure Pulse – Due to rock displacement at the face and the presence of low frequency waves

Figure 59 shows a low frequency wave from an air-blast during rock blasting at a coal mine;

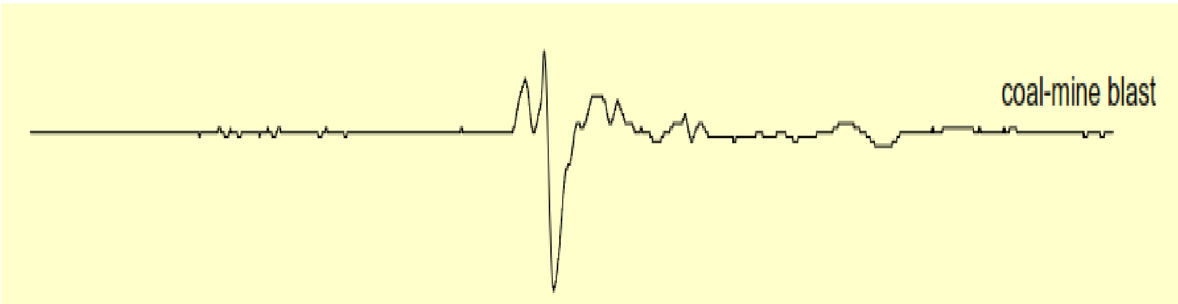

Figure 59 Low Frequency Wave from an Air Blast

Source: Best D., 2008

The following factors cause excessive air blasts;

- Lack of sufficient stemming length and Poor Quality of Stemming Material
  - When the stemming length is short, it is prematurely ejected during detonation
  - Poor quality stemming materials promote low frictional forces which are unable to withstand the high detonation pressure during detonation

- Escape of Gas Energy also known as Bubble Energy along the face wall
  - This may be due to existing rock fractures or due to lack of sufficient front-row burden

- Adverse weather conditions
  - High wind speed and wind direction
  - Temperature inversions especially in the morning

- Topographic features; not actually the cause but air blasts are heavily enhanced in down valleys than on higher grounds

- Surface detonation of detonating-cord trunk lines

- Inappropriate delay sequencing in the front face relative to the blast-hole spacing

7.5.2 Solution to Excessive Air-blasts

The following practices should be followed to control and manage excessive air blasts;

- Sufficient stemming length should always be used which must be at least 0.7 times the Burden
- Angular, crushed rock of appropriate size distribution in relation to the blast-hole diameter is highly recommended as the suitable good quality stemming material to be used during stemming
- Engineers should always check the free faces for excessive fractures from the back break and for the presence of voids and mud seams before blasting
- Engineers should always consult meteorological data before any blasting operation to check the weather of the day and should conduct the rock blasting in the afternoon because at this time of the day, there are few to none temperature inversions
- Blasting should be suspended at all times when there is adverse weather
- The use of non-electric shock tube initiation systems rather than detonating cords is highly recommended in order to prevent the surface detonation of trunk lines of the detonating cord

7.6 Excessive Dust

Figure 60 shows excessive dust being generated during a rock blasting operation, the image has been produced by editing a video of the blasting operation;

Figure 60 Video S16 Paused at 00:06:8.14 showing Excessive Dust during Detonation

(Further Details in Appendix)

Figure 61 shows dust travelling onto a residential area after rock blasting and it may travel faster depending on weather conditions of the day, for example when the wind speed is high and the wind direction faces the residential area;

Figure 61 Dust from a Blasting Operation moving towards a residential area

Source: Latest Mining News Magazine, 2014

7.6.1 Solution to Excessive Dust from Blasting

Rock blasting creates dust as the rock fragments due to the explosive energy and the dust can be very intense if the blasted rock contains silica, this can be a major problem in hard rock surface mines.

Research has shown that mines which have at least 8% Silica in their rocks are more likely to have large amounts of dust generated during blasting than mines which have a lesser percentage of Silica in their rocks (*Gazewood P., 2014*).

Dust is dangerous and harmful to workers who come in contact with it on an occasional basis, dust can cause short term illnesses and long term illnesses like *Silicosis* and that is why it is very important for engineers to be well equipped in managing and controlling dust at a mine site.

In order to control dust from rock blasting, the following practices should be applied;

- Spraying with clean water before blasting greatly helps in reducing the amount of dust generated. Clean water is important because if dirty water is used to spray the blast area, after evaporation the dirty water adds extra dust
- Water spraying also prevents settled dust from previous blasting becoming airborne
- Dust aprons should always be lowered during drilling
- Drills should be equipped with dust extraction cyclones
- Engineers should not use fine material from previous drilling operations as stemming material
- Engineers should ensure that sufficient material is used for stemming
- The Engineer in-charge should always ensure that blasting is only carried out after assessing the weather condition of the day, this ensures that wind speed and wind direction does not enhance dust emission from the mine site

Table 6 summarises dust control measures at a mine site;

Table 6 Dust Control Measures

Source: Gazewood P., 2014

DUST CONTROL METHOD	EFFECTIVENESS (Low is 10%-30%, moderate is 30%-50%, high is 50%-75%)	*COST AND DRAWBACKS*
Dilution ventilation	Moderate	High – more air may not be feasible
Displacement ventilation, including enclosure with extraction of dusty air	Moderate to high	Moderate – can be difficult to implement well
Wetting by sprays	Moderate	Low – too much water can be a problem
Airborne capture by sprays	Low	Low – too much water can be a problem
Airborne capture by high pressure sprays	Moderate	Moderate – can only be used in enclosed spaces
Foam	Moderate	High
Wetting agents	Zero to low	Moderate
Dust collectors	Moderate to high	Moderate to high – possible noise problems
Reducing generated dust	Low to moderate	Moderate
Enclosure with sprays	Low to moderate	Moderate
Dust avoidance	Moderate	Low to moderate

7.7 Presence of NO_X Gases during Blasting

Nitrogen is a component of many explosives used in the mining industry and during rock blasting because of the presence of oxygen during detonation and high temperatures involved NO_X gases are also generated.

Mono-Nitrogen Oxides (NO_X) gases are dangerous because they have adverse effects on human health and the environment, these gases are associated with respiratory diseases when inhaled, biological mutations when they react with other organic compounds and also they destroy the ozone layer in the stratosphere which is responsible for absorbing harmful UV light.

Figures 62 and 63 show NO_X gases during rock blasting, the images were taken at open-pit mine sites;

Figure 62 Yellowish Brown NO_X gases

Source: WordPress, 2014

Figure 63 NO_X gases spotted with a yellowish brown colour

Source: Ferret Australia, 2013

7.7.1 Causes and Solutions of NO_X during Rock Blasting

Table 7 Management of NO_X Gases during Rock Blasting

CAUSE	SOLUTION
Contamination of explosive column due to drill cuttings during loading	○ Ensuring that mine operators are efficiently trained ○ Minimising vehicle contact near blast holes ○ Using hole savers
Water Entrainment in the explosive	○ Removing top loading into wet blast holes ○ Sealing the top of the explosive column to prevent water ingress
Explosives formed by mixing two or more compounds, the inadequate or incorrect mixing results in NO_X emissions	Engineers should check the density of components when mixing, and should ensure mixing is done in accordance with manufacturer's standards
Degradation of the precursor during storage and transportation	○ Following recommended explosive storage and transportation procedures ○ The precursor should be checked and tested before blasting
Deterioration of the blast hole between drilling and loading i.e. cracks, voids, hole contraction	○ Ensuring that there is a short time between drilling and loading ○ Using hole savers ○ Drill hole stabilisation using mud that is compatible with the used explosive
Geochemistry of the rock i.e. high concentration of limestone	Examining geology of the area before choosing the appropriate type of explosive
High percentage of moisture in clay	○ Using water resistant explosives ○ If ANFO is still used then hole liners should be applied
Lack of confinement in soft rock and ground	○ Minimising the blast size
Mismatch between explosive type and rock type e.g. using a very powerful explosive on a soft rock	○ Examining rock mechanics before selecting explosive type
Contamination of the explosive with mud or sediment at bottom of the blast hole	○ Ensuring the primer is in an undiluted explosive ○ Using hole savers and gas bags to prevent contamination

7.8 Other Problems – Ecological

There may be situations when birds and animals are present in quarries and open-pit mines before the blasting operation.

Figures 64 and **65** highlight such a unique situation where birds were in an open-pit mine and were awaken by the rock blasting operation, it is shown in these two images from an actual blasting operation, birds flying away from the blast front;

Figure 64 Birds Flying Away from Blast Front

Source: **CLASSIFIED**

Figure 65 a Blast Front, Birds Flying Away

Source: **CLASSIFIED**

7.8.1 Proposed Solution

- Before every rock blasting operation, a physical inspection should be conducted around the quarry and the open-pit mine to see if there are birds and animals in the quarry and open-pit mine.

- If the inspection identifies that there are birds and/or animals in the quarry/open-pit mine, the animals and birds should be safely removed from the site before blasting and depending on local legislation appropriate government authorities can be contacted to remove animals and birds from the quarry/mine.

- An alarm with frequencies audible to birds and animals should be sounded before every rock blasting operation, this will also help in case during the physical inspection some birds and animals were missed.

8

OCCUPATIONAL SAFETY AND HEALTH DURING ROCK BLASTING AND EXPLOSIVES MANAGEMENT

Figure 66 Cartoon from UK Hazards Magazine

Source: Hazards Magazine Number 113, 2011

"Think Safety, Work Safely"

8.1 Introduction

The aspect of safety and health is much very important in every industry as it protects workers and allows them to perform their work efficiently and effectively by isolating scenarios which could lead to danger and harm. Occupational Safety and Health (OSH) is also called Occupational Health and Safety (OHS) or Work-Place Health and Safety (WHS).

OHS aims to ensure that workers carry out their duties in a safe and healthy environment, the same goal applies in the mining industry which includes rock blasting using explosives.

There are hazards involved when handling and using explosives for rock blasting, these may include; generation of hazardous fly rocks, generation of a huge amount of dust and toxic fumes during blasting, accidental fires caused by explosives, accidental detonation through misfires, inappropriate transport and storage practices.

Different legislation and standards exist in order to regulate and enforce OHS practices in the mining industry specifically in rock blasting with the use of explosives in open-pits.

The following bodies are involved in enforcing such measures; European Agency for Safety and Health at Work (**EU-OSHA**), the UK Safety and Health Executive (**HSE**) and the US Mine Safety and Health Administration (**MSHA**).

A lot of OHS principles discussed in this dissertation have been adopted from the UK HSE but some aspects will also incorporate information from the EU-OSHA and the US MSHA, a note will be placed in brackets when that is the case.

8.2 OHS Responsibilities of the Mine Manager

a) It is the responsibility of the Mine Manger to ensure that all workers at the mine site carry out their duties in a safe and healthy environment by enforcing the company's OHS policy

b) The Mine Manager should control, manage and supervise employees at the mine site, should also take responsibility in managing contractors and subcontractors engaged at the mine site

c) It is the task of the Mine Manager to ensure that;

   i. No person is able to work at the mine site without proper qualifications and training

   ii. No person is able to work in isolation with the risk of not being helped and/or discovered in time in case of an accident

   iii. No foreman shot firer (head blaster) is in charge if he is unable to supervise blasting workmen efficiently i.e. for example, when there are too many and scattered workmen to supervise at once

d) It is the duty of the Mine Manager to ensure that the whole mine adheres to mining safety regulations set by local, national and international authorities.

e) The Mine Manager should from day to day, consult with the safety and health personnel employed at the mine in order to identify practical measures including a change in organisational work plan and designs of safety systems so that the following points are met;

   i. Reduction and elimination of risks associated with mining operations

   ii. Control and management of identified risks

   iii. Recording of all identified hazards at the mine site with the intent of improving safety procedures and designs of safety systems

8.3 Open-Pit Mine Safety in Rock Blasting

The ***UK HSE's Health and Safety at Quarries; Quarries Regulation 1999; Approved Code of Practice and Guidance*** specifies the following in respect to occupational safety during rock blasting;

At the mine site there should be a blasting specification which caters to prevent and minimise the danger associated with rock blasting using explosives in the following way;

a) By ensuring that the risks associated with the projection of fly-rocks beyond the danger zone are low as practically and technically possible and the specification should outline ways of achieving this
b) Outlining ways of minimising misfires
c) By ensuring that faces are in a good and acceptable condition after each blast
d) The blasting specification should also incorporate information from previous blast works, accidents reports and blast design reports in order to effectively minimise hazards associated with rock blasting

It is very important that mine operators including shot-firers have competent skills in blasting and explosives, this minimises accident scenarios caused as a result of errors made by mine workers. Steps should be made by the company to check and improve the competence level of mine operators through, for example in-house training programmes.

It is also highly recommended for engineers to inspect the mine site, the face before and after each blasting operation, data collected during each inspection should be incorporated into the blasting specification to improve aspects of OHS for the next blast and future rock blasting operations.

The competence of the Mine Manger is vital as well as he/she supervises the staff working at the mine site, hence, Mine Mangers should become well acquitted with latest principles of OHS in Mining and Explosives Management.

8.4 Safety Aspects when Handling Explosives

a) If the mining company also operates an explosive magazine (storage facility), the company should take all the necessary measures to ensure that; containers containing electric detonators are isolated from any electric conductive surfaces, these containers should only be used to store detonators and the containers should be lined with protective material that is electric resistant, antistatic and able to absorb shock

b) Explosives should not be removed from manufacturer's packaging unless, they are about to be used or in special circumstances where close examination is required

c) In case explosives are directly transported to the mine site, the Mine Manger together with the Foreman Shot firer should check the delivery and ensure that everything is as it should be and explosives including accessories should never be left unattended, depending on the local jurisdiction, the Police or Military personnel are also present at the site

d) No one should smoke or have a naked light within 16 m of where explosives are being handled or stored

e) When transporting explosives, detonators and fuses should not be mixed and carried in the same container

f) During transportation of explosives either by rail or road to the mine site, nitro-compound and water-gel explosives should NOT be transported with other explosives

g) When transporting explosives by rail, other passengers are NOT allowed to board the train, only the Foreman Shot Firer and his assistants are allowed

h) When there are thunderstorms, the Foreman Shot Firer should suspend all blasting operations for that day and should ensure that no worker is left at the mine to prevent injury from accidental detonation

i) During charging of the blast-holes, there should be at least one and no more than two reliable workmen to assist the shot firer

j) It is the responsibility of the mining company through the Mine Manger and the Safety and Health personnel to ensure that mine operators including shot firers wear Personal Protective Equipment (PPE) at all times when working at the mine site
k) The PPE should be of good quality and construction, although the PPE does not offer 100% protection but when used together with other safety procedures and protocols ensures a much safer work environment at the mine site
l) Maximum care should be taken to prevent concussion and any mechanical stress to the explosives during transportation as this could cause an accidental explosion
m) When the mining company operates a magazine for storing Ammonium Nitrate, the following procedures should be followed;
   i. The ammonium nitrate should be contained in water-tight bags made of polyvinyl plastic or similar material
   ii. No stack of ammonium nitrate bags should contain more than 9000 kg or be higher than 2 m
   iii. The space between ammonium nitrate bags should be not less than 300 mm, and the space between the bags and the wall should also not be less 300 mm, and the space between stacks should not be less than 1m
   iv. No smoking including any open flame should be allowed in the magazine where ammonium nitrate is stored
n) After thorough inspection during rock blasting operations and an imminent risk is identified, the risk being prospective serious injury to mine personnel, the mining company’s safety and health policy must include immediate action to safeguard everyone at risk, actions may include; suspension of work in the area where the risk has been identified or sometimes the whole mine site, taking a faulty machine part or the whole plant out of use
o) The disposal explosive empty cases and deteriorated explosives is also one of the important safety aspect hence arrangements should be made to check that no explosive remains hidden in cases before disposal and sometimes manufacturers can advise on appropriate disposal practices for the particular explosive.

8.5 ESSEEM Safety Aspects in Rock Blasting

The ***European Shot firer Standard Education for Enhanced Mobility*** (ESSEEM) whose aim is to enhance education and training of shot firers all over across Europe has guidelines that promote safety practices in rock blasting and these safety procedures can today be applied in many mines across the world.

a) Mining Engineers should always be aware of the risk of drilling into explosives, risk of accidental explosion due to thunderstorms, lightning (static electricity), electromagnetic current, current leakage from high voltage power lines and electric powered machinery
b) Shot firers should never drill into holes that may have been previously charged
c) The mining company should ensure that shot firers have adequate training to understand the risk associated with accidental detonations and misfires
d) Engineers should always be aware of warning signals and must ensure that all safety procedures are met before explosives are fired for rock blasting
e) It is the responsibility of Mining Engineers and Shot Firers to ensure the following;
    i. The blasting specification is appropriate during the day of the blasting operation
    ii. The priming of cartridges and mixing of explosives are done in the correct and safest manner
    iii. Inspection and testing of shot firing circuit is done accurately
    iv. All possible misfires are checked accordingly and removed in the most safest way technically possible
    v. Shot firers working under the supervision of the Foreman Shot firer should mix explosives where needed, prime cartridges, charge and stem shot-holes, link and/or connect rounds of shots, withdraw persons and fire shots.

8.6 Current Statistics on Rock Blasting Accidents in the Mining Industry

There are many hazards and risks in the mining industry and blasting rock using explosives presents a grave danger when appropriate measures and procedures are not followed even though sometimes accidents do occur.

The Canadian Centre for Occupational Health and Safety (CCOHS) defines a **Risk** as "*the chance or probability that a person will be harmed or experience an adverse health effect if exposed to a hazard. It may also apply to situations with property or equipment loss.*"

And a **Hazard** as "*any source of potential damage, harm or adverse health effects on something or someone under certain conditions at work. Basically, a hazard can cause harm or adverse effects (to individuals as health effects or to organizations as property or equipment losses).*"

Current statistics on all mining accidents including rock blasting and explosives accidents occurring both in underground and surface mines will be examined from the United States, Canada, the United Kingdom, Australia, Portugal and the European Union (EU).

Most of the statistics show that safety aspects have greatly improved over the years in the mining industry as engineers now better understand risks and hazards associated with mining operations and are now equipped with advanced technologies to better design safety systems and safety policies, this was not the case during the early years of the 20th Century.

"*Think Safe, Work Safe*"

"*Think Safety, Work Safely*"

"*Safety First*"

8.6.1 United States

Data is from the ***US Department of Labor – Mine Safety and Health Administration (MSHA)***.

As of April 2014, 906 000 people were employed in the mining sector in the United States of which approximately 10 000 alone were added in April 2014 (*US Bureau of Labor Statistics*).

In 2011, mining and quarrying operations contributed 232 billion US dollars to the Gross Domestic Product (GDP) of the United States (*US National Mining Association (NMA*).

Table 8 shows the number of occupational injuries in the US mining sector from 2007 to 2013 together with the number of people that were employed from 2007 to 2013, data is for both surface and underground mines;

Table 8 Employment and Mining Injuries Statistics from US in Surface and Underground Mines

Source: US MSHA

	2007	2008	2009	2010	2011	2012	2013*
Number of Mines	14,871	14,907	14,631	14,283	14,176	14,093	13,708
Number of Miners	378,123	392,746	355,720	361,176	381,209	387,878	374,069
Fatalities	67	53	35	71	37	36	42
Fatal Injury Rate[1]	.0199	.0156	.0119	.0234	.0114	.0110	.0132
All Injury Rate[1]	3.43	3.25	3.01	2.81	2.73	2.56	2.46
Total Mining Area Inspection Hours/Mine[2]	44	56	59	63	62	61	59
Citations and Orders Issued[3]	144,081	173,555	173,100	170,117	156,521	139,156	118,759
S&S Citations and Orders (%)	29%	28%	30%	32%	30%	27%	27%
Dollar Amount Assessed (Millions)[4]	130.0	143.6	137.4	163.3	161.3	122.5	---

*Preliminary

Figure 67 shows a pie chart of **Fatal Injuries** involving different operations in the metal and non-metal mining industry including working with explosives for the year 2013 in the United States. Data is from both underground and surface mines.

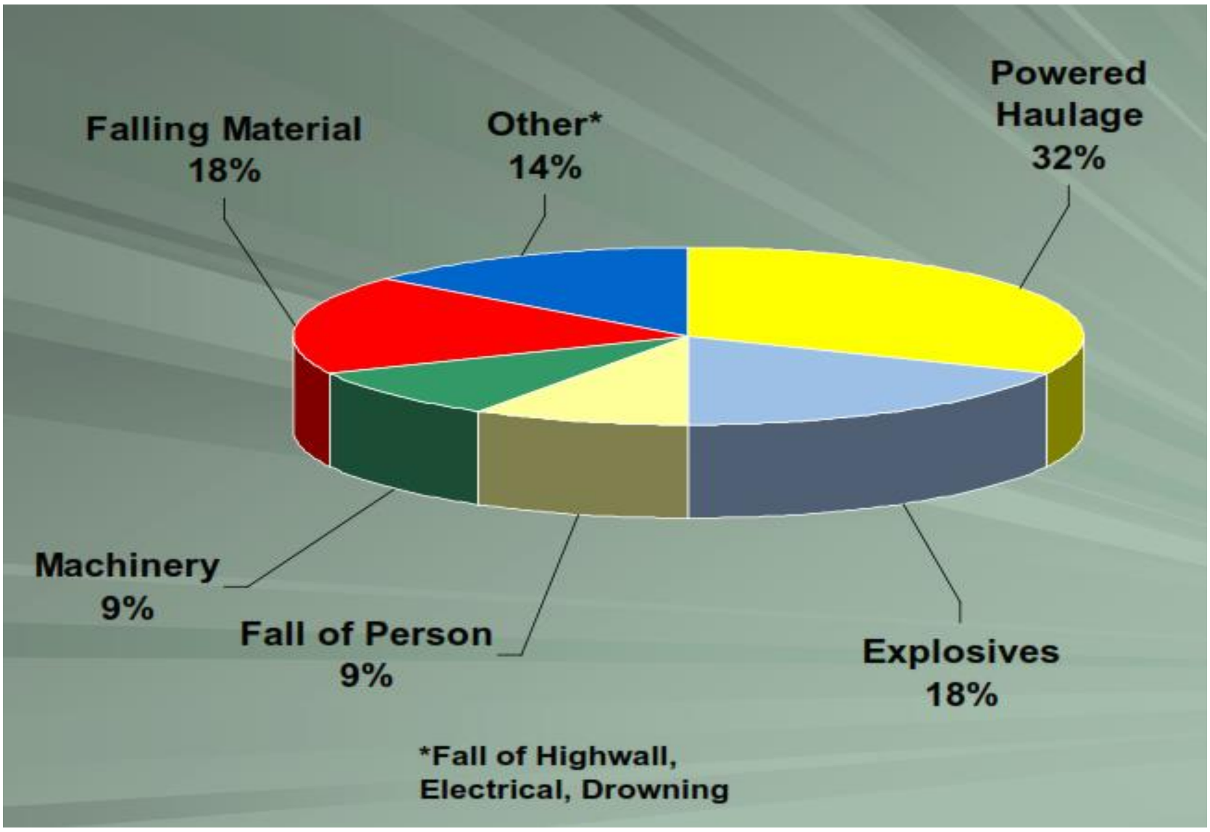

Figure 67 Metal and Non-Metal Mining Fatal Injuries by classification in the US in 2013

Source: US MSHA

Table 9 shows data for **Fatal Injuries** which occurred during accidents involving rock blasting using explosives in the US from 2008 to 2013.

Table 9 Fatal Injuries as a result of Accidents involving Explosives both in Underground and Surface Mines in the US

(Source: US MSHA)

Year	Fatalities	Number of Miners Employed
2008	0	392 746
2009	0	355 720
2010	1	361 176
2011	1	381 209
2012	0	387 878
2013	4	374 069

After the release of this data, the US Department of Labor – Mine Safety and Health Administration made the following recommendations;

- Mining companies should enforce safety procedures more strictly
- Supervisors must ensure that the blast area is cleared before each blasting operation
- Warning signals should be very audible before a scheduled blasting operation and mining companies should improve communication channels
- Mining companies should follow engineering procedures and try as much as technically possible to protect mine personnel from fly-rocks.
- Mining companies must have stringent ventilation and gas monitoring plans

Figure 68 shows a pie chart of **Fatal Injuries** by Occupation in the US for the year 2013. Data is both from metal and non-metal underground and surface mines.

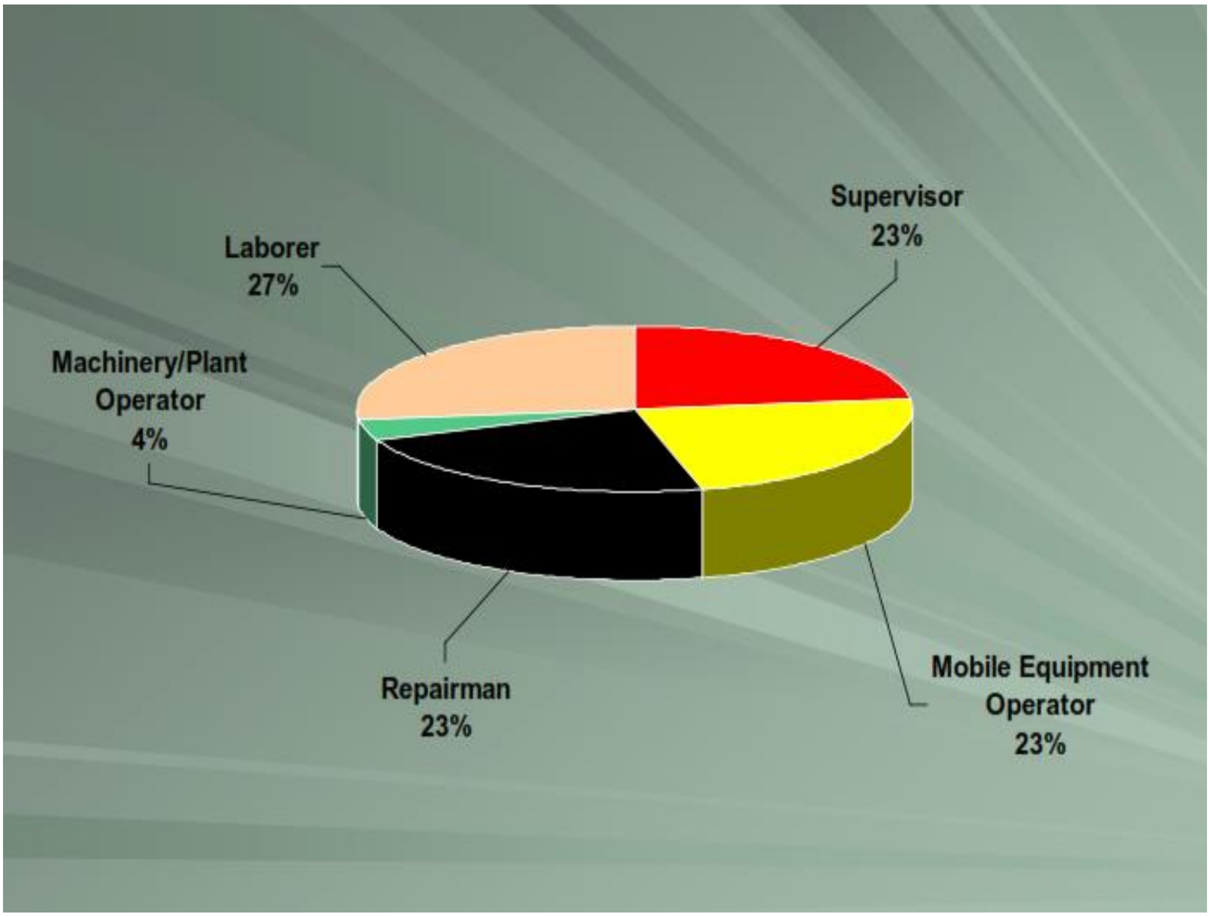

Figure 68 Metal and Non-Metal Mining Fatal Injuries by Occupation in the US in 2013

Source: US MSHA

For the year 2013, the US Department of Labor – MSHA, attributed the following as the main causes of fatal accidents in the metal and non-metal mining industry;

a) Failure to provide task training
b) Failure to conduct examinations
c) Failure to conduct Risk Assessments
d) Failure to conduct Pre-Operational checks
e) Failure to maintain equipment regularly
f) Failure to provide policies, procedures and controls
g) Failure to provide Personal Protective Equipment (PPE)

Figure 69 is a graph of fatal injuries in the US mining industry from 1978 to 2013, which occurred in surface and underground mines;

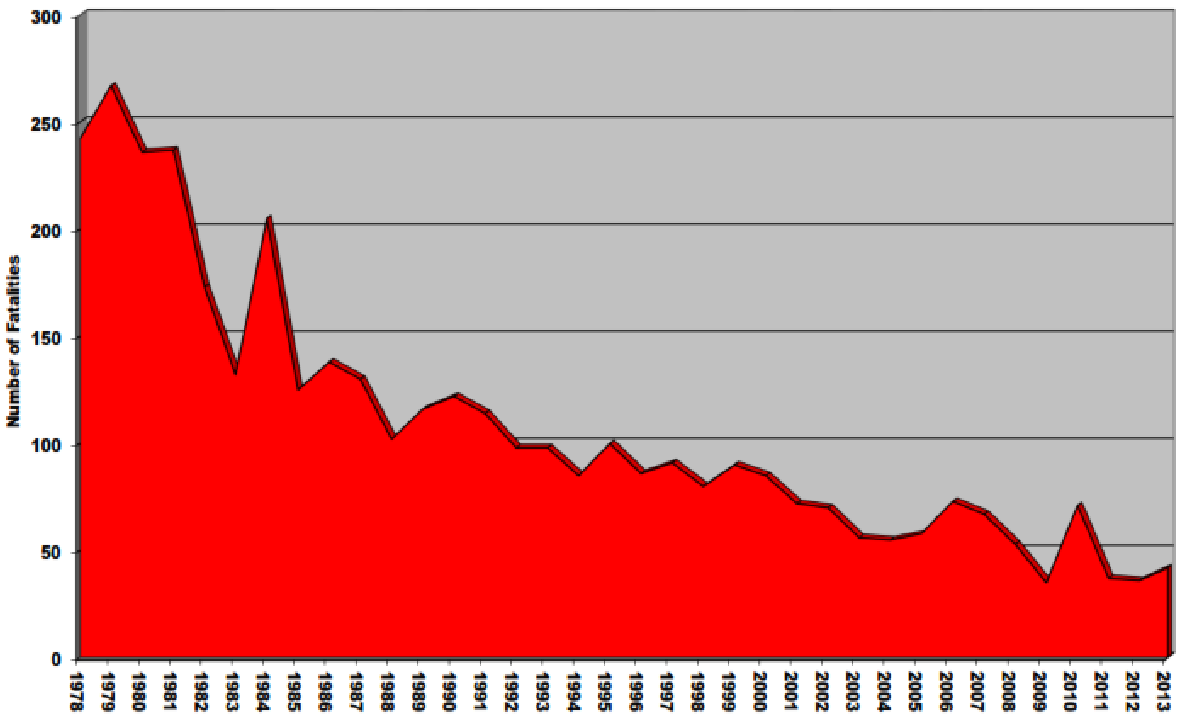

Figure 69 Fatal Injuries Graph from 1978-2013 in the US

Source: US MSHA

Table 10 shows Fatal Injuries in the Metal and Non-Metal Mining Industry starting from 1900 to 2012 in the United States. Data is from both underground and surface mines.

Table 10 Metal and Non-Metal Mining Fatal Injuries from 1900 – 2012 in the US

Metal/Nonmetal Fatalities for 1900 Through 2012

Please Note:
Metal/Nonmetal operations include Mills (Metal, Nonmetal, and Stone), Sand and Gravel, Surface (Metal, Nonmetal, and Stone), Underground (Metal, Nonmetal, and Stone).
Sand and gravel miners included starting in 1958.
Office workers at mine sites included starting in 1973.

Year	Miners	Fatalities	Year	Miners	Fatalities	Year	Miners	Fatalities	Year	Miners	Fatalities
1900	N/A	N/A	1930	N/A	376	1960	289,001	185	1990	235,690	56
1901	N/A	N/A	1931	159,007	225	1961	279,178	127	1991	230,107	53
1902	N/A	N/A	1932	116,079	139	1962	269,927	216	1992	224,567	43
1903	N/A	N/A	1933	125,347	159	1963	259,926	173	1993	219,320	51
1904	N/A	N/A	1934	138,689	181	1964	260,939	179	1994	225,498	40
1905	N/A	N/A	1935	177,160	222	1965	263,072	180	1995	229,536	53
1906	N/A	N/A	1936	193,957	308	1966	261,993	195	1996	229,045	47
1907	N/A	N/A	1937	217,020	310	1967	255,999	181	1997	235,915	61
1908	N/A	N/A	1938	192,567	247	1968	246,039	182	1998	235,561	51
1909	N/A	N/A	1939	203,843	232	1969	246,677	179	1999	238,852	55
1910	N/A	N/A	1940	213,619	307	1970	242,788	165	2000	240,450	47
1911	N/A	883	1941	226,220	322	1971	237,059	164	2001	232,770	30
1912	N/A	874	1942	210,409	361	1972	185,115	234	2002	218,148	42
1913	N/A	866	1943	183,726	314	1973	246,665	175	2003	215,325	26
1914	N/A	739	1944	158,282	237	1974	271,606	158	2004	220,274	27
1915	N/A	701	1945	145,637	174	1975	277,978	123	2005	228,401	35
1916	N/A	870	1946	162,408	181	1976	278,605	113	2006	240,522	26
1917	N/A	983	1947	174,586	220	1977	285,165	134	2007	255,187	33
1918	N/A	771	1948	176,364	203	1978	288,577	136	2008	258,918	23
1919	N/A	581	1949	182,638	152	1979	308,085	123	2009	221,631	17
1920	N/A	603	1950	180,955	164	1980	301,635	103	2010	225,676	23
1921	N/A	350	1951	185,244	175	1981	296,848	84	2011	237,772	16
1922	N/A	476	1952	186,463	209	1982	230,025	51	2012	250,228	16
1923	N/A	510	1953	188,692	161	1983	214,661	62			
1924	N/A	556	1954	177,425	139	1984	219,727	80			
1925	N/A	520	1955	184,239	157	1985	218,112	57			
1926	N/A	584	1956	200,807	172	1986	209,638	49			
1927	N/A	487	1957	219,151	152	1987	213,532	67			
1928	N/A	392	1958	269,076	167	1988	225,422	49			
1929	N/A	476	1959	288,560	173	1989	234,459	48			

Source: US MSHA

Table 11 shows Fatal Injuries in the Coal Mining Industry starting from 1900 to 2012 in the United States. Data is from both underground and surface mines.

Table 11 Coal Mining Fatal Injuries from 1900 – 2012 in the US

Coal Fatalities for 1900 Through 2012

Please Note:
Office workers included starting in 1973.

Year	Miners	Fatalities	Year	Miners	Fatalities	Year	Miners	Fatalities	Year	Miners	Fatalities
1900	448,581	1,489	1930	644,006	2,063	1960	189,679	325	1990	168,625	66
1901	485,544	1,574	1931	589,705	1,463	1961	167,568	294	1991	158,677	61
1902	518,197	1,724	1932	527,623	1,207	1962	161,286	289	1992	153,128	55
1903	566,260	1,926	1933	523,182	1,064	1963	157,126	284	1993	141,183	47
1904	593,693	1,995	1934	566,426	1,226	1964	150,761	242	1994	143,645	45
1905	626,045	2,232	1935	565,202	1,242	1965	148,734	259	1995	132,111	47
1906	640,780	2,138	1936	584,582	1,342	1966	145,244	233	1996	126,451	39
1907	680,492	3,242	1937	589,856	1,413	1967	139,312	222	1997	126,429	30
1908	690,438	2,445	1938	541,528	1,105	1968	134,467	311	1998	122,083	29
1909	666,552	2,642	1939	539,375	1,078	1969	133,302	203	1999	114,489	35
1910	725,030	2,821	1940	533,267	1,388	1970	144,480	260	2000	108,098	38
1911	728,348	2,656	1941	546,692	1,266	1971	142,108	181	2001	114,458	42
1912	722,662	2,419	1942	530,861	1,471	1972	162,207	156	2002	110,966	28
1913	747,644	2,785	1943	486,516	1,451	1973	151,892	132	2003	104,824	30
1914	763,185	2,454	1944	453,937	1,298	1974	182,274	133	2004	108,734	28
1915	734,008	2,269	1945	437,921	1,068	1975	224,412	155	2005	116,436	23
1916	720,971	2,226	1946	463,079	968	1976	221,255	141	2006	122,975	47
1917	757,317	2,696	1947	490,356	1,158	1977	237,506	139	2007	122,936	34
1918	762,426	2,580	1948	507,333	999	1978	255,588	106	2008	133,828	30
1919	776,569	2,323	1949	485,306	585	1979	260,429	144	2009	134,089	18
1920	784,621	2,272	1950	483,239	643	1980	253,007	133	2010	135,500	48
1921	823,253	1,995	1951	441,905	785	1981	249,738	153	2011	143,437	21
1922	844,807	1,984	1952	401,329	548	1982	241,454	122	2012	137,650	20
1923	862,536	2,462	1953	351,126	461	1983	200,199	70			
1924	779,613	2,402	1954	283,705	396	1984	208,160	125			
1925	748,805	2,518	1955	260,089	420	1985	197,049	68			
1926	759,033	2,234	1956	260,285	448	1986	185,167	89			
1927	759,177	2,231	1957	254,725	478	1987	172,780	63			
1928	682,831	2,176	1958	224,890	358	1988	166,278	53			
1929	654,494	2,187	1959	203,597	293	1989	164,929	68			

Source: US MSHA

8.6.2 Australia

Data is from the ***Australian Bureau of Statistics*** and ***Safe Work Australia***, a governmental agency that seeks to develop and collaborate national OHS policies across all Australian territories (including Northern Territory, Western Australia, Victoria, New South Wales, South Australia, and Queensland).

Data from ***the Chamber of Minerals and Energy of Western Australia*** has also been used from 1970 to 2006.

In 2012, 276 300 people were employed in the Australian mining industry which represented 2.4% of all the Australian workforce at the time (*innovation.gov.au*).

From 2011 to 2012 mining and quarrying contributed 130.75 billion US dollars to the GDP of Australia (*innovation.gov.au*).

Table 12 shows data of fatal injuries which occurred in the mining sector from 2003 to 2012, fatalities are from metal and coal mining accidents both in underground and surface mines.

Table 12 Fatal Injuries in the Mining Sector in Australia from 2003 to 2012

(Source: Safe Work Australia.gov.au and Australian Bureau of Statistics)

Year	Number of Fatalities	People Employed
2003	11	84 240
2004	11	87 167
2005	8	101 900
2006	15	128 200
2007	7	135 900
2008	12	137 700
2009	10	173 900
2010	5	165 900
2011	5	245 000
2012	7	276 300

The Government in Western Australia (WA) also collected data on fatalities from 1970 to 2006 within the mining sector as shown in **figure 70**; it shows fatal injuries from different mining operations including using explosives to fragment rock in both underground and surface mines.

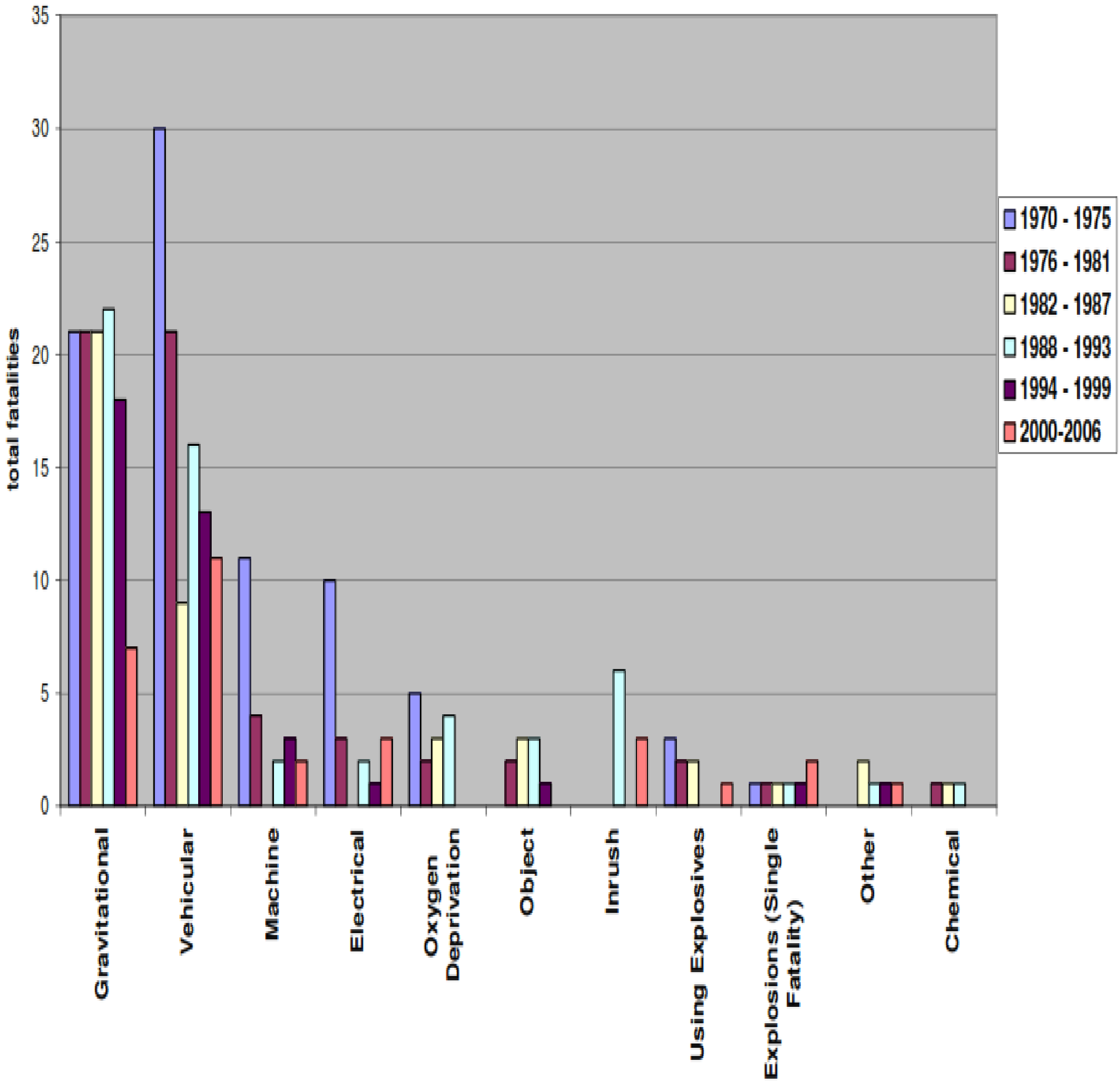

Figure 70 Fatalities in WA from 1970 – 2006 both in Surface and Underground Mines

Source: Chamber of Minerals & Energy of Western Australia, 2006

In total there were 306 mining fatal injuries in WA from 1970 to 2006.

Figure 71 shows mining fatal injuries in WA in 6-year intervals from 1970 to 2006 both in surface and underground mines;

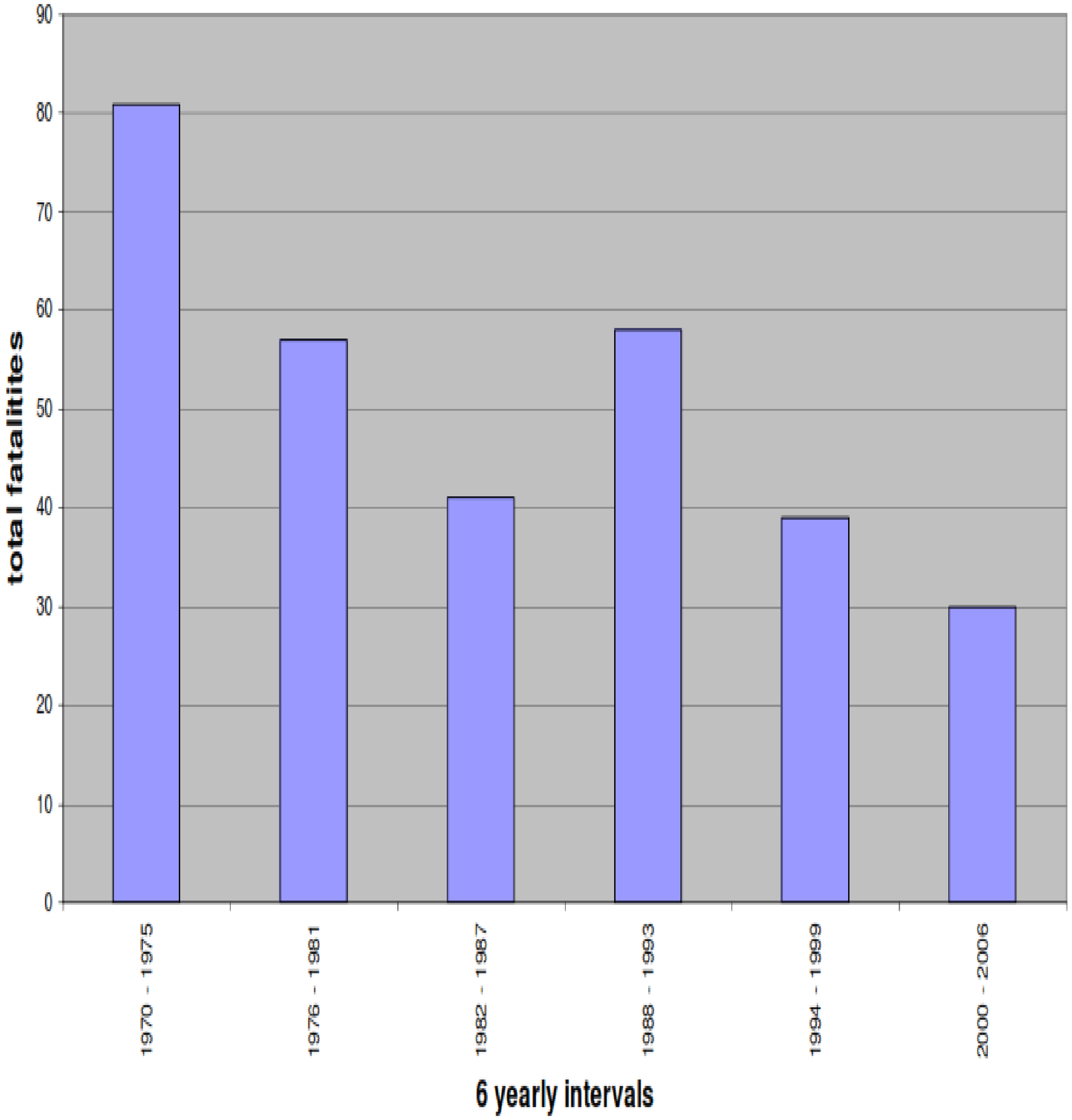

Figure 71 Six-Year Interval Fatal Injuries in WA from 1970 – 2006 both in Surface and Underground Mines

Source: Chamber of Minerals & Energy of Western Australia, 2006

8.6.3 Portugal

According to the Energy and Geology Directorate (***Direcção Geral de Energia e Geologia (DGEG)***) mining and quarrying operations in Portugal contributed 1.56 billion US dollars to the GDP in 2011.

Data is from the Ministry of Social Security and Solidarity (***Ministério da Solidariedade e da Segurança Social***) and the Portuguese Government Statistics Database (***Base de Dados Portugal Contemporâneo – PORDATA)*** and the DGEG.

Table 13 shows the number of fatal injuries in the mining industry that includes surface and underground mines in Portugal from 2000 to 2012;

Table 13 Number of Fatal Injuries in Surface and Underground Mines in Portugal from 2000 to 2012

(Source: DGEG, PORDATA and Ministério da Solidariedade e da Segurança Social)

Year	Number of Fatalities	People Employed
2000	9	12 160
2001	16	11 470
2002	5	12 370
2003	8	10 758
2004	12	10 624
2005	6	10 257
2006	3	9 943
2007	4	8 966
2008	5	8 864
2009	8	8 325
2010	5	10 005
2011	5	10 100
2012	4	9 430

8.6.4 The European Union (EU)

Data is from the ***Eurostat***; a European Union (EU) Statistics Database.

In 2006, 732 200 people were employed in the mining industry within the EU and mining operations in the European Union contributed 327.9 million US dollars to the total EU GDP at that time (*Eurostat*). Data from the ***European Statistics on Accidents at Work*** (ESAW) has also been used from 2008 to 2009. **Table 14** shows **Non-Fatal Injuries** in the mining sector from 2008 to 2009 in the EU;

Table 14 Non-Fatal Injuries in EU Mining Sector; both in surface and Underground Mines

	Number	Number	Number		Incidence rate	Incidence rate	Incidence rate
	2008	2009	2010		2008	2009	2010
Belgium	229	172	151		8273	5665	4944
Bulgaria	251	179	176		651	472	538
Czech Republic	979	813	663		1782	1559	1384
Denmark	96	79	77		2116	1739	1707
Germany	2932	2861	2794		2292	2125	2390
Estonia	72	81	69		1196	1271	1005
Ireland	89	39	65		873	545	916
Spain	5330	4495	3818		13882	13852	12222
France	1031	847	528		4068	2905	2214
Italy	1170	960	930		3272	2897	2604
Cyprus	17	24	19		3409	3358	2933
Latvia	12	8	7		391	266	221
Lithuania	15	7	7		383	270	294
Luxembourg	19	16	11		5689	4878	3448
Hungary	85	75	80		:	760	718
Malta	37	23	9		12242	6166	1871
Netherlands	:	:	531		:	:	6338
Austria	517	272	186		4884	4599	3047
Poland	2889	3012	3114		1236	1252	1741
Portugal	1525	1076	1174		8512	6031	5822
Romania	326	341	301		306	340	464
Slovenia	172	154	153		4958	4780	5186
Slovakia	:	16	:		:	148	:
Finland	168	132	129		2488	2942	2095
Sweden	118	90	97		1275	1193	1236
Great Britain	651	684	614		500	683	766
Norway	523	575	373		1292	1291	699
Switzerland	195	215	225		7178	5193	7846

(Source: ESAW)

Table 15 shows both **fatal and non-fatal injuries** from 2008 to 2010 within the European Union, data combines both underground and surface mining operations;

Table 15 Fatal and Non-Fatal Injuries in the EU Mining Sector from 2008 – 2010

(Source: ESAW)

UNIT	Number	Number	Number		Incidence rate	Incidence rate	Incidence rate
EU27	2008	2009	2010		2008	2009	2010
fatal	116	105	80		13	12	11
non-fatal	18730	16457	15703		2069	1873	2148

Table 16 shows the top 5 EU member states with the largest number of people employed in the mining industry in 2006;

Table 16 Top 5 EU Countries with the largest labour force in Mining and Quarrying in 2006

(Source: Eurostat)

Country	People Employed	% of Total Employed in EU-27 Mining Sector (%)
Poland	188 600	24.4
Romania	134 300	17.4
Germany	87 600	11.9
United Kingdom	65 600	9
Czech Republic	44 400	6.1

8.6.5 United Kingdom

In 2006, mining and quarrying operations contributed 47.72 billion US dollars to the GDP of the United Kingdom at that time. (*UK Office for National Statistics*).

Data is from the ***Health and Safety Executive*** (HSE) and the ***UK Office for National Statistics***.

Table 17 shows fatal injuries in the United Kingdom which occurred from 1995/1996 to 2002/2003 and then from 2007/2008 to 2010/2011, the data is from both coal and metal surface and underground mines;

Table 17 Occupational Fatal Injuries in the UK both in Surface and Underground Mines, Coal, Metal and Quarrying Operations

(Source: UK HSE)

Year Interval	Number of Fatalities	People Employed
1995 – 1996	0	93 700
1996 – 1997	2	87 800
1997 – 1998	1	82 900
1998 – 1999	3	88 200
1999 – 2000	0	78 900
2000 – 2001	0	76 250
2001 – 2002	1	78 300
2002 – 2003	0	69 000
2007 – 2008	2	65 600
2008 – 2009	1	63 000
2009 – 2010	3	62 000
2010 – 2011	1	65 000
2011 – 2012	2	73 000

Figure 72 shows employment in the coal mining industry from 1880 to 2012 in the United Kingdom, employment figures are in thousands;

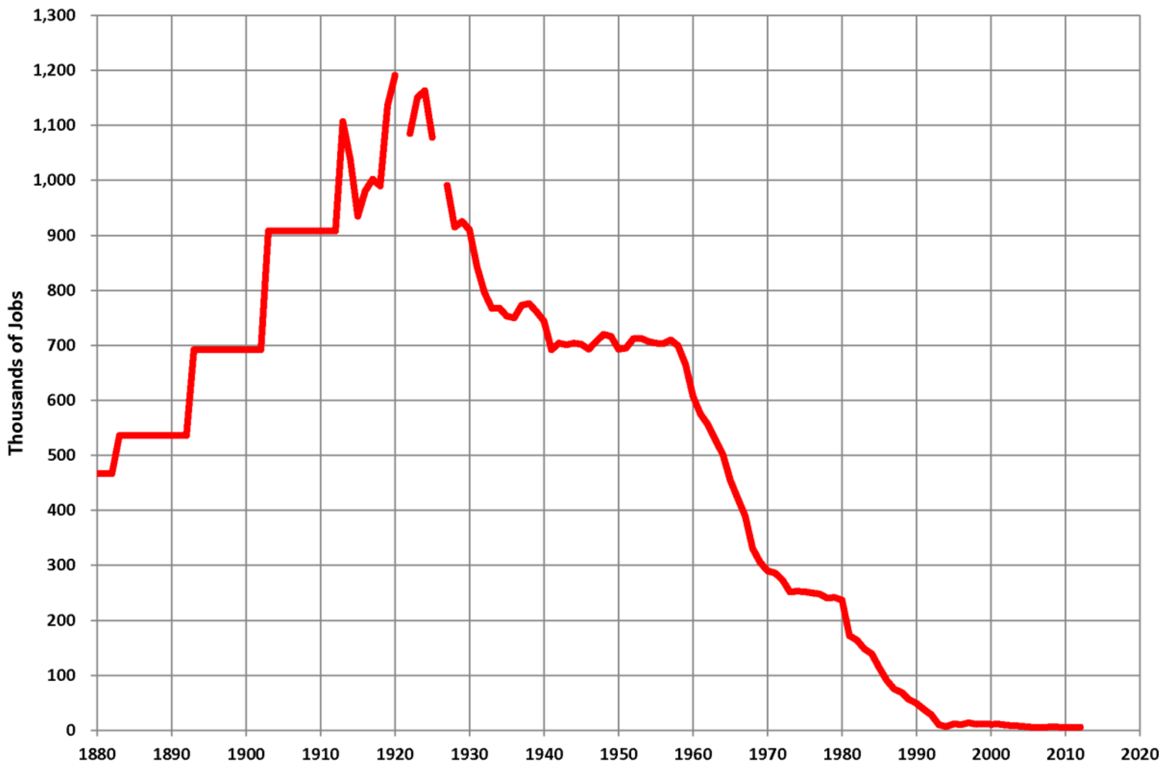

Figure 72 Employment in the UK in Coal Mining Industry

Source: Wikipedia

8.6.6 Canada

The ***Mining Association of Canada*** (MAC) carried a survey and found that in 2010 mining and quarrying operations contributed 33.1 billion US dollars to the GDP of Canada.

308 000 people were employed in the Canadian mining sector in 2010 (*the Mining Association of Canada*).

Data is from the ***Government of Canada Labor Program*** (***Canadian Ministry of Labor***) and was gathered from 2002 to 2007 from the *Employer's Annual Hazardous Occurrence Report* (EAHOR) provided by every federal jurisdiction in Canada.

FTE – Full Time Equivalents

DIIR – Disabling Injury Incidence Rate; it is the number of disabling and fatal occupational injuries per 100 employees expressed as full time equivalents (FTEs)

IIR – Injury Incidence Rate; this is the measure of all occupational injuries (fatal, disabling and minor) per 100 employees and expressed as full time equivalents (FTEs)

FIIR – Fatal Injury Incidence Rate; the number of total fatal occupational injuries per 100 000 employees expressed as full time equivalents (FTEs)

Table 18 shows mining occupational injuries in Canada from 2002 to 2007 both in underground and surface mines across all territories of Canada.

Table 18 Occupational Injuries in the Mining Sector in Canada from 2002 – 2007

Year	**Injuries**				**Employment (FTEs)**		**Total Hours Worked**	**Injury Rates**		**Fatality Rate**
	Minor	Disabling	Fatal	**Total**	Office	Total		DIIR	IIR	FIIR
2002	62	5	0	67	124	596	1,148,279	0.84	11.24	0
2003	67	5	0	72	132	571	1,022,546	0.88	12.61	0
2004	23	0	0	23	214	763	1,534,892	0	3.01	0
2005	83	2	0	85	265	927	1,921,358	0.22	9.17	0
2006	105	6	0	111	313	1307	2,458,477	0.46	8.49	0
2007	117	7	0	124	269	703	1,436,453	1	17.64	0

(Source: Government of Canada Labor Program)

8.6.7 STATISTICS SUMMARY

By looking at the injury statistics in the mining industry in the US and Australia, a lot of accidents that do happen in mines are not due to the use and handling of explosives, this is also true not only in the US and Australia but across the globe, a lot of occupational injuries that occur in different mining operations both in surface and underground mines are due to powered haulage, vehicular energy (for example, being hit by a truck) and use of plant and machinery.

In the EU in 2006, Poland had the largest number of people employed in the mining industry which represented 24.4 % of all people employed in the mining sector in EU-27 and in that same year, Spain had the largest number of non-fatal mining occupational injuries.

9

OPTIMISING THE BLASTED MATERIAL THROW

9.1 Introduction

The process of optimising the blasted material throw is built from trajectory theory which is used to predict the horizontal throw, the aspect of using Trajectories to predict the throw was developed by Workman et al. (*Moore A.J., 2005*).

The model assumes the following factors;

- That effect of rock dimension and density on throw is not that significant when the throw is between 0 – 300 m (0 to 985 ft.)
- That effect of air resistance and wind direction on the predicted throw is not that significant when the throw is in the range of 0 to 300 m (0 – 985 ft.)
- *Workman et al* found that this model is accurate enough for throw distances up to 300 m, however for distances above 300 m, the ignored factors become very significant and hence contribute to error in the prediction, hence at $>$ 300 m, factors such as air resistance, rock geometry, rock density, throw momentum are very crucial in predicting the throw of the blasted material and for these situations more complex models exits which incorporate all these factors.

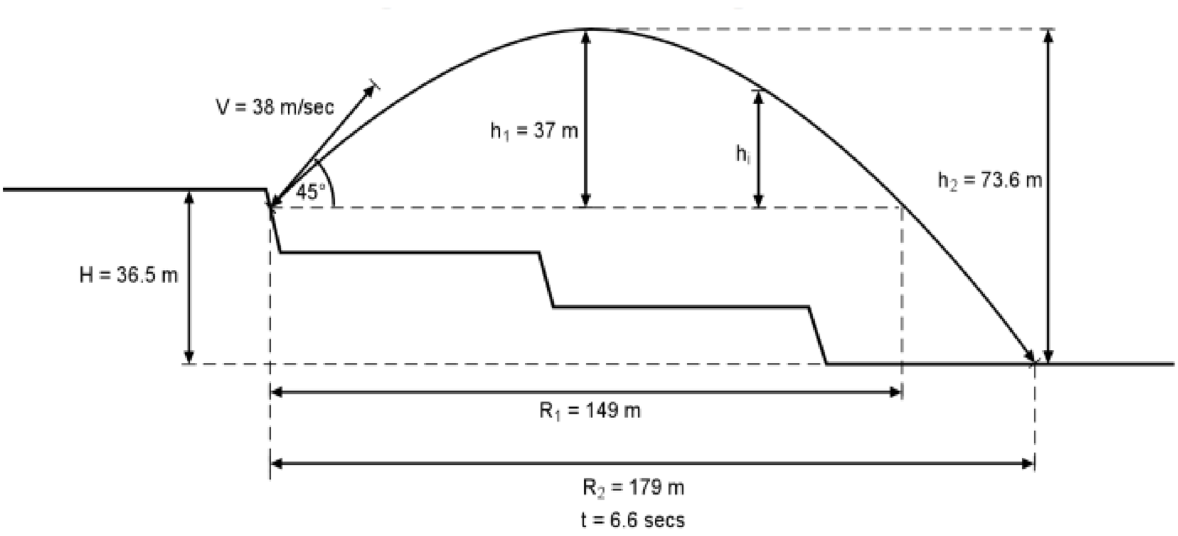

Figure 73 Throw Trajectory Model

(Workman et al, 1994)

$$\boldsymbol{L = V_0\, Cos\, \theta \left(\frac{V_0\, Sin\, \theta + \sqrt{(V_0\, Sin\, \theta)^2 - 2gH}}{g}\right)}$$

$$\boldsymbol{V_0 = k\left(\frac{\sqrt{m}}{SH}\right)^{1.3}}$$

L – Horizontal Throw of Blasted Material (m)

V_0 – Face Launch Velocity (m/s)

θ – Launch Angle (degrees)

m – Explosive charge mass per Delay (kg)

H – Elevation (m)

g – 9.81 m/s^2 (acceleration due to gravity)

k – Constant (depends on type of explosive and mineral content of the rock)

SH – Stemming Height (m)

9.2 Practical Application of the Trajectory Throw Model

As mentioned earlier $\boldsymbol{k\ constant}$ depends on the mineral content of the rock whether it is sulphide mineral or oxide mineral and also depends on the type of explosives used.

Sulphide Minerals contain the sulphide anion (S^{2-}) as the major ion, examples of sulphide minerals are; Pyrite (FeS_2), Chalcocite (Cu_2S), Nickeline ($NiAs$) and Pyrargyrite (Ag_3SbS_3).

Oxide Minerals contain oxide anion (O^{2-}) as the major ion, examples of oxide minerals are Columbite ($(FeMnNbTa)_2O_6$), Cuprite (Cu_2O), Corundum (Al_2O_3) and Uraninite (UO_2).

When ANFO is used;

$k = 21.9$ (Sulphide Zone Blast)

$k = 28.3$ (Oxide Zone Blast)

When ENERGAN explosive is used;

$k < 17.2$ (Sulphide Zone Blast)

$k < 22.3$ (Oxide Zone Blast)

In the example which aims at demonstrating the practical application of the model, ANFO will be used to blast rock containing oxide minerals hence $\boldsymbol{k = 28.3}$

NB. Because all blasted material lands on the ground floor due to gravity the elevation (H) is always = 0, **hence for practical calculations H = 0 m**, however the model can be used to accurately predict the throw at any elevation within the throw range of 0 to 300 m.

Given the following data; (m = 3, 6, 9, 12, 15, 18, 21, 24) kg

H = 0 m (ground floor)

$\theta = 45^0$

$k = 28.3$

$g = 9.81$ m/s^2

SH = 4 m

Then using MATLAB to calculate the Blasted Ore Throw with all other factors constant and changing only the Charge mass per Delay of the explosive (m), I created a MATLAB Function with the following code;

```
function throw(k,m,SH,x,H)

% V = Face Launch Velocity (m/s)

% x = Launch Angle (Degrees)

% Throw = Metres


V=((sqrt(m))/SH)^1.3*k

g=9.81;

Throw=((V*cos(degtorad(x)))*((V*sin(degtorad(x)))+sqrt(V^2*sin(degtorad(x)).^2-
2*g*H)))/g
```

The MATLAB code above creates a MATLAB function **throw(k,m,SH,x,H)** for calculating the Blasted Ore Throw in metres and the Face Launch Velocity (V) in m/s, **k** = constant, **m** = explosive charge mass per delay (kg), **SH** = stemming height (m), **x** = launch angle (degrees), and **H** = elevation (m).

After calculation using the MATLAB function the following results shown in **Table 19** were obtained;

Table 19 Explosive Charge Mass per Delay vs Blasted Ore Throw

Charge Mass Per Delay (kg)	Face Launch Velocity (m/s)	Blasted Ore Throw (m)
3	9.53	9.26
6	14.96	22.81
9	19.47	38.64
12	23.47	56.17
15	27.14	75.07
18	30.55	95.15
21	33.77	116.26
24	36.83	138.3

From these results in **Table 19**, it can be shown that by increasing the explosive charge mass, the throw of the blasted ore also increases with all other factors constant.

With this kind of data Mining Engineers are able to select the Optimum Charge per Delay in order to get the Optimum Blasted Ore Throw.

Having the Optimum Blasted Ore Throw is very important because it means a save in resources at the mine site, for example, an optimum throw removes the need of trucks to travel a longer distance to collect the blasted ore for further processing hence saving fuel and also eliminates the danger of fly-rocks reaching outside the set blast area.

The graph in **Figure 74** was produced using MATLAB from the results obtained in **Table 19** and it supports the observation that by holding all other factors constant and increasing the Charge Mass per Delay of ANFO also increases the throw of the blasted ore and vice versa.

In fact increasing the Charge Mass per Delay of an explosive means increasing the explosive energy that is delivered to the rock mass.

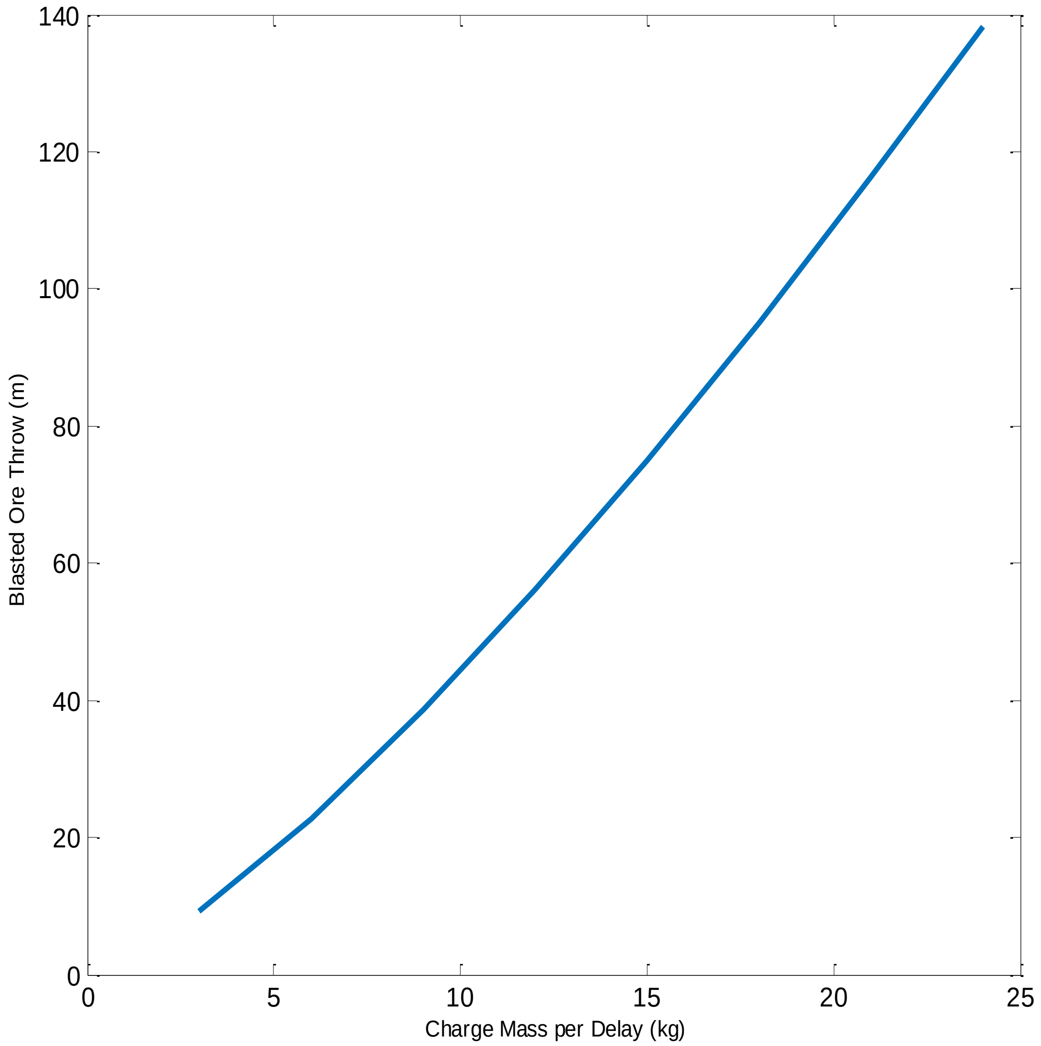

Figure 74 Charge Mass per Delay vs Blasted Ore Throw

Source: Graph Produced using MATLAB

10

CONCLUSION

It has been shown from this dissertation that a lot of problems associated with rock blasting in open-pit mines have to do with inappropriate blast design, this makes blast design a very significant parameter for engineers to consider before any rock blasting operation.

There is a relationship between the causes of blasting problems in relation to blast design parameters, for example, it has been shown that faulty stemming leads to the escape of explosive energy but the escape of explosive energy is also the cause of excessive fly-rocks and air-blasts, this means one problem may lead to extra two problems. Another example has to do with insufficient delay which leads to excessive fly-rocks but also causes excessive air-blasts.

In this dissertation, two parameters have come out as important variables in controlling the blast throw and hence excessive fly rocks; **Explosive Charge per Delay** and **Stemming Height**, by choosing the optimum Explosive Charge per delay and Stemming Height, the Blast Throw can be optimized and fly rocks can be reduced because by lowering the explosive charge per delay the blast throw and fly rock distance is reduced and by increasing the stemming height the blast throw is also reduced together with fly rock distance.

The Risk Analysis Model for misfires assumes that there is a misfire, in other words a misfire must occur during blasting, but if there are no misfires then there is no problem associated with misfires this means that the greatest way to control misfires is preventing them from occurring hence engineers should try as practically as possible to prevent the occurrence of misfires.

10.1 Recommendations for Future Work

The following work is recommended as future work;

- Examine Detonation Failure and Over-fragmentation in surface mines using video image analysis
- Develop a guideline of recording rock blasting operations in surface mines using high speed video cameras
- Carry out Fly-rock Throw Field Tests to Prove the Authenticity of Existing Fly-rock Throw Mathematical Models
- As a general recommendation, research work in reducing fly-rocks from rock blasting operations should continue because even today, the danger associated with fly-rocks is very high and fly-rocks still remain as the most dangerous aspect of rock blasting in the mining and construction industry.
- Examine the effects of rock properties (compressive strength, elasticity, fractures, degree of weathering, in-situ stress, tensile strength, discontinuities, shear strength) on the degree of rock fragmentation during blasting
- Study the applications of Fracture Mechanics in rock fragmentation during blasting
- Investigate the role of Ergonomics in Mine Safety and Mining Equipment Design
- Examine the effectiveness of engineering controls in underground mines in relation to Occupational Safety and Health

11

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APPENDIX A

Figure 75 VIDEO S8 (Port Hedland, Australia) Paused at 0:0:56.04

Source: YouTube

Figure 76 VIDEO S8 (Port Hedland, Australia) Paused at 0:0:56.2

Source: YouTube

Figure 77 VIDEO S8 (Port Hedland, Australia) Paused at0:0:56.26

Source: YouTube

Figure 78 VIDEO S8 (Port Hedland, Australia) Paused at 0:0:56.7

Source: YouTube

Figure 79 VIDEO S1 (Canada) Paused at 0:0:57

Source: YouTube

Figure 80 VIDEO S13 (British Columbia, Canada) Paused at 0:0:4.13

Source: YouTube

Figure 81 VIDEO S16 (Washington State, United States) Paused at 0:0:17.02

Source: YouTube

Figure 82 VIDEO S16 (Washington State, United Sates) Paused at 0:2:46.2

Source: YouTube

Figure 83 VIDEO S16 (Washington State, United States) Paused at 0:7:57.1

Source: YouTube

Figure 84 Bench Blasting in South Africa

Source: Benco Blasting, 2014

Figure 85 Escape of Explosive Energy

(Photograph by Naoya Hatakeyama)

Figure 86 Open-pit Rock Blasting

(Photograph by Naoya Hatakeyama)

Table 20 Visual NO_X Rating Scale

Source: Australian Explosives Industry and Safety Group Inc., 2011

Level	Typical Appearance
Level 0 No NOx gas	
Level 1 Slight NOx gas 1A Localised	
1B Medium	
1C Extensive	
Level 2 Minor yellow/orange gas 2A Localised	
2B Medium	
2C Extensive	
Level 3 Orange gas 3A Localised	
3B Medium	
3C Extensive	
Level 4 Orange/red gas 4A Localised	
4B Medium	
4C Extensive	
Level 5 Red/purple gas 5A Localised	
5B Medium	
5C Extensive	

Assessing the amount of NOx gases produced from a blast will depend on the distance the observer is from the blast and the prevailing weather conditions. The intensity of the fume produced in a blast should be measured on a simple scale from 0 to 5 based on the table above. The extent of the fume also needs to be assessed and this should be done on a simple scale from A to C where:-

A = Localised (ie Fume localised across only a few blast holes)

B = Medium (ie Fume from up to 50% of blast holes in the shot)

C = Extensive (ie Extensive generation of fume across the whole blast)

Table 21 Field Colour Chart for Visually Identifying NO_X from Rock Blasting

Source: Australian Explosives Industry and Safety Group Inc., 2011

Level	Colour	Pantone Number
Level 0 No NOx gas		Warm Grey 1C (RGB 244, 222, 217)
Level 1 Slight NOx gas		Pantone 155C (RGB 244, 219, 170)
Level 2 Minor yellow/orange gas		Pantone 157C (RGB 237, 160, 79)
Level 3 Orange gas		Pantone 158C (RGB 232, 117, 17)
Level 4 Orange/red gas		Pantone 1525C (RGB 181, 84, 0)
Level 5 Red/purple fume		Pantone 161C (RGB 99, 58, 17)

APPENDIX B

Table 22 Video Library

Name	Location	Web Source	Year	Web Address	Video Duration (min. sec.)	Extra Information
S1	United States	YouTube	2009	URL removed on YouTube	1:04	N/A
S2	Australia	YouTube	2011	http://www.youtube.com/watch?v=7fEJcyMNfII	03:45	N/A
S3	Kalkkimaa, Finland	YouTube	1964	https://www.youtube.com/watch?v=EpxzScvkcM0	03:57	N/A
S4	Pilbara, WA, Australia	YouTube	2011	https://www.youtube.com/watch?v=UbEgAw0uZkQ	03:10	Iron Ore Mine Brockman 2 Mine owned by Rio Tinto
S5	Newman, WA, Australia	YouTube	2010	https://www.youtube.com/watch?v=AlNoQoR2-fs	06:24	Mount Whaleback Iron Ore Mine, owned by BHP Billiton
S6	Mongolia	YouTube	2005	URL removed on YouTube	00:19	N/A
S7	Canada	YouTube	2011	https://www.youtube.com/watch?v=0c3tVpRP1_c	00:17	N/A
S8	Port Hedland, Australia	YouTube	2012	https://www.youtube.com/watch?v=y6qZHSIAz70	02:11	Iron Ore Mine, owned by BHP Billiton http://www.bhpbilliton.com

Table 23 Video Library Continuation…

Name	Location	Web Source	Year	Web Address	Video Duration (min. sec.)	Extra Information
S9	Switzerland	YouTube	2011	https://www.youtube.com/watch?v=rCljPCkSm10	04:24	N/A
S10	Queensland, Australia	YouTube	2010	URL removed on YouTube	00:39	N/A
S11	Queensland, Australia	YouTube	2009	http://www.youtube.com/watch?v=myqj_TCnIvA	00:31	N/A
S12	Canada	YouTube	2007	http://www.youtube.com/watch?v=Bj55eMVXFQU	00:46	N/A
S13	British Columbia, Canada	YouTube	2010	http://www.youtube.com/watch?v=3uXDhwsWppI	04:30	N/A
S14	Queensland, Australia	YouTube	2011	http://www.youtube.com/watch?v=s6lkrBX3Dgg	07:40	N/A
S15	Scotland, United Kingdom	YouTube	2013	http://www.youtube.com/watch?v=NkxKX8Z6vhY	00:54	50 000 kg Explosive: 200 000 Tonnes Granite. Glensanda Super Quarry
S16	Washington State, United States	YouTube	2012	https://www.youtube.com/watch?v=OoyVe1eDlT8	14:16	McCallum Rock Drilling Inc. http://mccallumrockdrilling.com/